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METALS

THEIR PROPERTIES AND TREATMENT

BY

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PARTIALLY REWRITTEN AND AUGMENTED

BY

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NEW EDITION

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PREFACE TO NEW EDITION.

OWING to the important changes which have taken place in the metallurgy of *iron* since this work was first published, it has been found necessary to re-write entirely the portion relating to that subject. In doing this, the opportunity has been taken to break down the unnatural barrier which in text-books usually exists between iron and steel. In the comparatively limited space available for the purpose it has been attempted to give a clear and consecutive account of the principles upon which this industry, of unsurpassed importance, is based.

Since the successful extraction and treatment of metals, for the most part, depends on the nature and supply of *fuel*, and *refractory materials*, no hesitation has been felt in introducing chapters on these subjects into the present edition. No book on metallurgy would be complete without them.

The greater importance which *nickel* and *cobalt* have recently assumed fully warrants the insertion of the few pages concerning them.

The remaining alterations and additions do not call for special comment. It will doubtless be thought by some that many recent processes which have not been mentioned—the greater part only modifications of existing methods—should have received notice.

Many a process which is heralded with great trumpetings to-day, in two or three short years gives place to some other equally pretentious claimant, and is heard no more of. So must it ever be. I do not flinch from taking the responsibility of all such omissions, holding, as I do, that nothing, until it has stood the test of time, should find place in a work of this nature.

For the illustrations to the part on *gaseous fuel*, and for that of the Cleveland blast-furnace, I am indebted to Dr. C. W. Siemens, and to Messrs. E. A. Cowper, W. Whitwell, and Windsor Richards respectively. The following illustrations are from electrotypes kindly furnished by the publishers of the works from which they are taken, viz.: Coke-ovens (*Chemistry as applied to the Arts and Manufactures*); ordinary puddling-furnace and Danks' furnace (*Bauermann's Metallurgy of Iron*); Dormoy's rabble (*Encyc. Brit.*).

The neat diagrams of the Pernot furnace were made by my friend Mr. W. E. Koch, from drawings with which the inventor was so good as to furnish me.

Valuable assistance has been rendered me by Mr. W. Terrill in the revision of *copper*.

In conclusion, my acknowledgments are due to my assistant, Mr. W. G. McMillan, for the care he has bestowed on the revision of the proof sheets, and on the making of the index, as well as for his aid in compiling from various sources the tables at the end of this book.

A. K. HUNTINGTON.

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METALS:

THEIR PROPERTIES AND TREATMENT.



FUEL

THE word FUEL comes, through the low Latin *focale* and Norman-French *fuayl*, from the Latin *focus*, a hearth or fire-place, and means, therefore, anything which can be burnt on the *hearth* with generation of heat.

Heat is due to molecular motion, and as in the production of heat from fuel molecular motion is not induced by mechanical action, as in the case of friction, it follows that some other agency must be at work. The motive force in this case is termed *chemical affinity*: the force which, under suitable conditions, unites certain dissimilar bodies together into a single homogeneous substance. Thus, if the elementary substance carbon, the largely predominating constituent of our principal fuel, be heated to a certain point in contact with atmospheric air, union takes place between the carbon and the oxygen, with evolution of heat and the production of a homogeneous gaseous compound. The maximum combining power of carbon with oxygen is one of carbon with two of oxygen, represented symbolically by the formula CO_2 . Combination to this extent having taken place, the maximum amount of heat obtainable from the so-called combustion of carbon has been produced.

All that is essential in order that a substance may with

reason be termed a fuel is that it can be practically employed for the generation of heat.

From this point of view, many substances which the advance of science has caused to be utilised for the production of heat in various metallurgical processes should be classed as fuel ; and, in order to obtain a proper appreciation of the subject, it is essential that it should be so. Thus, *sulphur* in the minerals iron and copper pyrites is in certain cases just as much a fuel as the carbon in coal, since it is capable of uniting with oxygen, evolving at the same time very considerable heat, which can be utilised. Similarly, *silicon* and *phosphorus* play an important part in the production of heat in certain processes, which we shall have to consider later on in treating of the manufacture of iron and steel.

In the ordinary acceptance of the word, fuel is composed entirely of vegetable matter, which has, at some time or other, grown on the surface of the earth. Vegetable matter is composed of carbon, hydrogen, and oxygen, with smaller proportions of nitrogen and earthy matter.

Although there is no means of estimating the *absolute* quantity of heat evolved by the combustion of a body, yet the *relative* quantities of heat given out during the combustion of different bodies may without difficulty be determined. This is done by finding the number of parts by weight of water which one part by weight of the substance will, on perfect combustion, raise one degree in temperature. In this way it has been substantiated that one part by weight of carbon, in the form of purified wood-charcoal, in uniting with $2\frac{2}{3}$ parts by weight of oxygen to form carbonic acid (CO_2), in which state carbon is oxidised to the maximum, evolves heat sufficient to raise 8,080 parts of water one degree. In the same way it may be shown that the unit of hydrogen, in uniting with 8 parts by weight of oxygen to form water (H_2O), would raise 34,462 parts by weight of water one degree. For purposes of comparison it is important that water of approximately the same initial temperature should

be employed, for at different temperatures water does not require the same amount of heat to raise it one degree. In considering the *useful effect* which can be obtained from the combustion of hydrogen allowance must be made for the heat required to maintain the product of combustion in the state of vapour. This being done, the number 29,161 is obtained instead of 34,462.

The heat-evolving power of a substance during its combustion will be spoken of as its *calorific* power. The absolute amount of heat given out by a substance in burning is the same whether the combustion takes place rapidly or slowly, but the *intensity* will be greater in the one case than in the other. This is a very important point in considering the value of a fuel for any given purpose. The calorific intensity of a fuel will depend partly on the state of its aggregation and freedom from moisture, partly on its composition and the nature of the ultimate products of combustion. The products of combustion are either gases or vapours, and as, according to their composition, they will have different specific heats, the temperature obtainable must vary with the composition of the fuel and the way in which it is burnt.

Thus, we have seen that the calorific power of hydrogen is 34,462, but the product of combustion of hydrogen is vapour of water, and it therefore becomes necessary, in considering the intensity of the heat produced, to deduct the heat required to maintain the water in the vaporous condition. For one part of water this amounts to 537 heat units. Therefore, in the combustion of one part of hydrogen $9 \times 537 = 4,833$ heat units are occupied in keeping the product in the state of vapour. In the estimation of the calorific power it will be understood that this vapour is condensed, and the so-called latent heat becomes sensible, and does work in the calorimeter by melting ice or raising the temperature of water as the case may be. In calculating the intensity of combustion we have further to take into account the specific heat of steam ($=0.4805$), and also the difference between the

specific heat of water and of steam. For the sake of simplicity the initial temperature is assumed to be 0°C. , and the barometric pressure 760 millimetres of mercury. The pressure is also assumed to be constant.

Then, T being the temperature to be ascertained, we have

$$T = \frac{34,462 - \{ [(1 - 0.4805) \times 100] + 537 \} \times 9}{9 \times 0.4805} = 6,743^{\circ}\text{C.}$$

The calorific power of carbon we have seen to be 8,080. The specific heat of carbonic acid, as determined by Regnault, is 0.2164. There is an important distinction between this case and that which we have just considered: the product of combustion is not condensable at ordinary pressures, and we have, therefore, in making our calculation, nothing to do with its latent heat; and further, the products of combustion in the case of hydrogen are relatively heavier than those of carbon, and they are also of a higher specific gravity. 3.6 parts by weight of carbonic acid are produced by the combustion of 1 part of carbon, therefore

$$T = \frac{8,080}{3.6 \times 0.2164} = 10,183^{\circ}\text{C.}$$

The essential difference between calorific power and calorific intensity should be sufficiently clear from the foregoing calculations. Roughly speaking, the calorific power of carbon is to that of hydrogen as 3 : 13, whereas their relative intensity is in the ratio of 3 : 2. The advantage of a high percentage of carbon in fuel to be employed for the generation of high temperatures must also be apparent, as well as the importance of using dry fuel.

Although the foregoing considerations sufficiently express the *difference* in value of various substances in respect to their heat-evolving power, and the *relative* intensity of that heat, yet, for reasons to be explained, the numbers we have so far obtained for the intensity do not accurately represent the state of things which exists in practice. In the first place we have supposed the combustion to have been

effected by pure oxygen. When atmospheric air is employed, which would ordinarily be the case, the quantity of nitrogen, carbonic acid, and moisture present, and their specific heats, have to be taken into account. For the sake of simplicity we will assume that nitrogen and oxygen are alone present in the air to be used ; then, as in air for every part of oxygen there are 3.35 parts by weight of nitrogen, the nitrogen to be heated during the conversion of one part of wood-charcoal into carbonic acid will amount to $3.35 \times 2.6 = 8.93$ — 2.6 being, it will be remembered, the quantity of oxygen required to burn one part of carbon. The specific heat of nitrogen is 0.244. Introducing these figures into the equation, we have

$$\frac{8,080}{(3.6 \times 0.2164) + (8.93 \times 0.244)} = 2,718^{\circ} \text{C.},$$

instead of $10,183^{\circ} \text{C.}$ The presence of carbonic acid and moisture would still further reduce the temperature. In the same way it may be shown that, instead of $6,743^{\circ} \text{C.}$ in the case of hydrogen we should only get $2,684^{\circ} \text{C.}$ when atmospheric air (free from carbonic acid and moisture) was used. The temperature may be reduced in many other ways, notably by the carbon not being oxidised to the maximum.

So far we have only considered the oxidation of carbon to carbonic acid, which takes place when a suitable quantity of air is supplied to the fuel, and other conditions are favourable ; but when this is not the case, a lower oxide of carbon, known as carbonic oxide (CO), may be formed. The formation of this oxide takes place when carbonic acid is passed through red-hot carbon : thus, $\text{CO}_2 + \text{C} = 2\text{CO}$. Suppose, for the sake of example, that a tall upright cylinder be filled with red-hot fuel, and that air be admitted at the bottom beneath the fuel, then carbonic acid will be produced where the atmospheric oxygen comes in contact with the fuel ; as, however, the carbonic acid traverses the next layer of hot fuel it will be reduced to carbonic oxide, as shown in the equation, and in this state it will remain until it reaches the

top of the cylinder, and there comes in contact with a supply of air, when it will combine with oxygen and reform carbonic acid. Now, it is clear that the temperature in the cylinder will be that which would result from the direct formation of the lower oxide from carbon, since whatever is given out during the first formation of carbonic acid is re-absorbed during the subsequent reduction of the carbonic acid to carbonic oxide. It remains, then, to ascertain the calorific power and the intensity of the heat of carbon when burnt to carbonic oxide. This cannot be arrived at by direct experiment, but by ascertaining the amount of heat evolved during the conversion of carbonic oxide into carbonic acid, and deducting that amount from the heat evolved by carbon when burnt direct to carbonic acid, the required information is obtained. By experiment, the heat given out by carbonic oxide whilst burning to carbonic acid has been found to be 2,403 heat units. The amount of carbon in carbonic oxide $= \frac{1}{2} \frac{2}{8} = \frac{3}{7}$ of its weight. Therefore, to obtain the same unit as that employed in calculating the calorific power of carbon, it is necessary to multiply by $\frac{7}{3}$; thus, $\frac{7}{3} \times 2,403 = 5,607$ are the number of heat units obtained by the combustion of carbonic oxide to carbonic acid. But carbon in burning to carbonic acid evolves 8,080 units, therefore, in the production of carbonic oxide, $8,080 - 5,607 = 2,473$ heat units only become sensible, *i.e.* less than one-third that which results from complete oxidation. If, instead of in oxygen, carbon were burnt to carbonic oxide in air (free from CO_2 and H_2O) then the calorific power would be only 1494°C ., the specific heat of carbonic oxide being taken as 0.245. The intensity would be $1,222^\circ \text{C}$. The fact, that so much less heat is generated by the union of carbon with oxygen to form carbonic oxide than during the oxidation of the latter to carbonic acid, is to be explained by the circumstance that in the first case a solid body passes into the gaseous condition, and in the accomplishment of this transition energy is absorbed which otherwise would have taken the form of heat.

These facts are of the highest interest from an economic point of view. The few figures just considered show better than any words could do the enormous loss which results, when fuel is burnt without suitable arrangements for a supply of air, in proper proportion and in such a way that the heat generated can be utilised to the greatest advantage. The introduction of an excess of air is nearly as objectionable as an insufficient supply, for the excess has to be heated at the expense of the heat of combustion, and the intensity is proportionately reduced.

The calorific intensity or thermal effect of carbonic oxide burnt in oxygen is $7,073^{\circ}\text{C.}$, and in air $2,982^{\circ}\text{C.}$ It will be observed that in the latter case the thermal effect is greater than in the case of carbon, though in the former it is less. This result is traceable to the difference in calorific power, and the weight of the products of combustion in the two cases.

For the sake of comparison, the initial temperature of combustion has been assumed to be 0° in all cases. It is manifest that whatever heat is possessed by the fuel and air before combustion must assist in the ultimate intensity obtained. The figures which represent this initial temperature should, therefore, be incorporated in the numerator of the equation when the actual temperature in any particular case is required to be known. Within the last fifty years great progress has taken place in blast-furnace practice, in consequence of advantage having been taken of this means of increasing the available heat of combustion ; and within the last twenty years a new industry, the manufacture of so-called soft steel in the Siemens' furnace, has sprung into existence from a similar cause.

There remains one more means of increasing the intensity of combustion. It will be remembered that, besides assuming the initial temperature to be 0° , the pressure was taken as constant at 760 mm., *i.e.* the gases as they were heated were free to expand. If the pressure be increased,

the maximum temperature at any given moment will also increase. A convenient way of understanding the difference in the two cases will be to consider the products of combustion at constant volume, instead of at constant pressure.

If a given quantity of gas be raised in temperature t degrees, the pressure being kept constant, a certain increase in the volume will have taken place. Now let the gas be compressed until it has its original volume, and its temperature will be found to have been increased by the mere act of compression. The *specific heat at constant volume* is less, then, than the *specific heat at constant pressure*. The ratio between them, based on the theory of gases is,*

Air . . .	1 : 1.409	Hydrogen . . .	1 : 1.413
Oxygen . . .	1 : 1.402	Carbonic acid . . .	1 : 1.257
Nitrogen . . .	1 : 1.420	Carbonic oxide . . .	1 : 1.411

In practice, the difficulty of working under pressure is very great. In all cases, however, where a blast is employed the pressure in the immediate vicinity of the point at which the blast is introduced, and which is also the point of maximum temperature, must be greater than that of the atmosphere.

Where, then, is the limit to the temperature to be obtained by heating the fuel and the oxygen or air required for its combustion, and by working under pressure? The limit is closer at hand than at first might have been anticipated. In the single word *dissociation* we have the key-note to the answer to this question. If a gas at 0° , consisting of two or more different atoms, be maintained at a constant pressure, and its temperature be raised, at a certain point the force which retains the atoms in union will be overcome, and a portion of the gas will be split up into its constituents in order to restore the tension at which the particular gas is stable. By reducing the pressure the temperature at which dissociation commences is lowered. H. Sainte-Claire Deville compares the dissociation of a gas below the tem-

* Jamin, *Cours de Physique*, 1878, vol. ii. p. 74.

perature of its formation to the evaporation of volatile substances below their boiling points, and he defines the tension of dissociation and vapour tension in exactly the same way. Thus, he found that the vapour of water commences to undergo dissociation between $1,100^{\circ}$ and $1,300^{\circ}$ C. Yet it has been shown that the ignition of a mixture of hydrogen and oxygen in equivalent proportions, and at the ordinary atmospheric temperature, gives a temperature of about $2,500^{\circ}$ C. Similarly, carbonic acid partially decomposes into $\text{CO} + \text{O}$ between $1,100^{\circ}$ and $1,200^{\circ}$ C.

At a pressure of nine to ten atmospheres, Bunsen found that carbonic oxide in uniting with oxygen was capable of producing a temperature of $3,000^{\circ}$ C., as against $2,600^{\circ}$ to $2,700^{\circ}$ obtainable at the ordinary pressure of the atmosphere. Similar experiments with hydrogen gave a temperature of about $2,850^{\circ}$, as compared with $2,500^{\circ}$ C. at the ordinary pressure. When air instead of oxygen is employed, according to Bunsen, carbonic oxide, burnt under a pressure of seven or eight atmospheres, produces a temperature of about $2,000^{\circ}$ C., and hydrogen about the same. These figures correspond to about $1,700^{\circ}$ to $1,800^{\circ}$ C. at the ordinary pressure, and therefore differ somewhat from those obtained by Deville's method. In either case they suffice to indicate the principles involved, and in that sense are sufficient for our purpose. There are great difficulties in the way of determining with exactitude these very high temperatures.

At about the ordinary atmospheric pressure, Deville found that carbonic oxide began to dissociate into carbon and oxygen at a bright-red heat. From considerations based on the temperature of the sun, Bessemer some years ago proposed to construct a furnace to work under pressure. Up to the present, no important advance has been made in this direction ; it is a subject not likely, however, to be lost sight of. Improvements of this nature can result only from the application of mechanical contrivances by skilled engineers.

From the foregoing considerations it is not difficult to determine the relative useful effect obtainable from hydrogen, carbon, and carbonic oxide respectively, when these substances are burnt separately. When, however, a complex body, containing carbon, hydrogen, and oxygen, has to be dealt with, the difficulty is very greatly increased. In the case of a solid, such as wood or coal, the hydrogen and oxygen, which we have hitherto been considering as gases, are in the solid condition, and their state of union with the carbon and with one another is to a great extent unknown. It thus often happens that calculations based on the assumption that the constituents of any given fuel are arranged in some preconceived way, and that during combustion they will be given off in some particular state of combination, give results wide from those obtained in practice. Two fuels may perfectly well have the same elementary composition, yet the molecules may be differently constituted, and the proportion of volatilisable matter in the two cases will not be the same. In consequence of this, it has been necessary to fall back upon generalisations derived from a number of experiments on fuels of different composition. In this way it has been substantiated that the calorific power in the case of coal and lignite increases and decreases with the proportion of solid residue or coke produced. This is, however, only true within certain limits, which do not include anthracite coal and bituminous lignites; the former being excluded on account of the excess of carbon they contain, the latter on account of their excess of hydrogen. In order, then, to judge fairly of the nature and value of a coal it should be submitted to experiment, to determine the relative proportion of volatile and solid matter it will yield when heated with exclusion of air, and also what kind of coke is produced, *i.e.* whether it is compact and hard, or light and friable. In this way more reliable information will be obtained than by making an analysis to determine the relative proportion of the ultimate constituents.

WOOD.—Wood consists almost entirely of cellulose, a body which has the following percentage composition and to which the formula $C_{12}H_{20}O_{10}$ has been assigned.

Composition of Cellulose.

Carbon	44.44
Hydrogen	:	:	:	:	:	:	6.17
Oxygen	49.39
							<hr/> 100.00

The other organic constituents vary with the species of tree ; they play a part in determining the degree of inflammability of the wood—*e.g.* the ease with which pine inflames is due to the resin it contains. The value of wood as a combustible is, however, mainly dependent on its density and dryness.

The following represents the average composition of dry wood, exclusive of ash : carbon 51, hydrogen 6.24, oxygen 41.5, nitrogen 1. In different species the hydrogen and nitrogen do not sensibly vary, and the carbon and oxygen differ generally not more than about 1 per cent. from the figures given. Unless in exceptional cases the ash ranges between 1 per cent. and 2 per cent. The ash consists mainly of potash and lime and smaller proportions of soda and magnesia, oxides of iron and manganese, in combination with carbonic acid mainly, but to a small extent also with hydrochloric acid, sulphuric acid, and phosphoric acid. The soluble portion of the ash amounts to about 15 to 20 per cent.

It will readily be seen from the composition of wood that it is unsuitable for use where it is required to produce a high temperature. The calorific power of cellulose has been experimentally determined to be about 3,600. An additional reason for not employing wood in metallurgical operations is to be found in the fact that it retains, even after prolonged air-drying, 18 to 20 per cent. of moisture. The desiccation of wood is greatly promoted by the removal of the bark. The younger branches of trees contain, as a rule,

about twice as much water as the older and the trunk ; the quantity of moisture in either case being least at the fall and the beginning of the year, and greatest in the spring when the leaves and shoots begin to form. By air-drying wood shrinks to the extent of $\frac{1}{10}$ per cent., or more. Wood inflames at a temperature of about 300° C. in the presence of oxygen or air.

It will be observed that in wood there is a large proportion of oxygen present, and as it is in chemical combination with the hydrogen and carbon, it follows that a given weight of wood cannot produce the same amount of heat as would an equal weight of a substance composed of carbon and hydrogen alone. For all practical purposes it is as if the carbon and hydrogen had already been partially burnt, and the products of combustion remained mixed with the unburnt wood. When wood is converted into CHARCOAL by being strongly heated, with, as far as possible, the exclusion of air, these compounds are given off as gases and vapours, and the residue consists almost entirely of carbon, a little hydrogen and oxygen being retained even under the most favourable conditions. Wood is best cut for charcoal when it is about thirty years old. The volatile products of the distillation consist mainly of water, wood spirit, pyro-ligneous acid, oils and tar, and, towards the end principally, carburetted hydrogen gas, carbonic oxide, and carbonic acid. The process of distillation commences at about 150° C. and is complete at about 360° to 400° C. The nature and relative quantity of the volatile products will be dependent mainly on whether the heat is slowly or rapidly increased, and on its initial intensity. A glance at the result of submitting wood to distillation is sufficient to make it clear that, except considerable local heat is required, it would be more economical to dry the wood only before use, unless the products of the distillation can be collected and utilised.

Good charcoal preserves the form of the wood from which it is made. Its density will depend to some extent

on that of the wood employed, but also to a considerable extent on the way in which the carbonisation has been conducted. If very rapidly, the charcoal may be light, fissured, and friable; if slowly, compact and dense. A high temperature also conduces to this latter result as well as to the more complete expulsion of the gaseous elements, which, however, at the highest temperature we can command, are never entirely got rid of by the action of heat alone. When properly made charcoal should be perfectly black, hard, and sonorous, and it should not soil the fingers when touched. If the charcoal does not answer to this description, either the temperature has been too low, or it has been partially burnt by undue admission of air. In the latter case it will be of a dull appearance, tender and friable, and more or less cracked, and will mark the fingers.

Charcoal, like artificially dried wood, absorbs moisture from the air, the extent to which this takes place depending on the temperature at which the charcoal has been produced, and on its density. These latter conditions also determine the temperature at which combustion will commence. When the charcoal has been prepared below about 400° C., the temperature required to ensure its ignition is greater than that at which it was prepared; above 400° C. it is less. Thus, if prepared at 400° C. it would inflame at about 370° , but if prepared at the fusing-point of steel then the temperature required to start combustion would be about 600° to 800° C. For most purposes it is not desirable to prepare charcoal at a higher temperature than 360° C.

Preparation of Charcoal.—What is known as charcoal-burning is carried on sometimes by building the wood into a stack or pile in the open air, in a way hereafter to be described; sometimes by placing the wood in closed chambers of brick, stone, or iron, and subjecting it to the action of heat applied externally, and which is occasionally obtained by the combustion of the volatile products of the distillation. When the charring is effected in piles in the

open air, a dry and level piece of ground, sheltered from the wind, and, if possible, near a water supply, should be chosen. In a circular pile the bed on which the pile is to rest should be made to slope upwards slightly towards the centre. Three stakes are driven into the bed at its centre ; round these are piled pieces of wood, which in some districts are placed vertically and concentrically in two or more layers, in others horizontally, a few pieces round the chimney only being in a vertical position. In either case the top is covered over with smaller pieces, laid horizontally and concentrically, so as to bind the whole

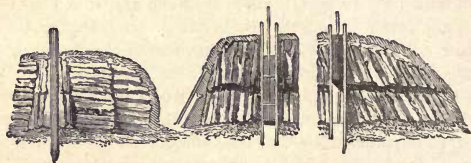


FIG. 1.

together. All interstices are also filled up, as far as possible, with small wood. Next a row of forked sticks is driven into the ground at the base of the pile, and a layer of branches placed on the top of them to support the cover, which is then constructed of turf, the grassy side placed inwards. Instead of these sticks and branches, stones are sometimes used. The cover is completed by plastering over the turf with a mixture of *breeze*, *i.e.* small refuse charcoal, and earth, or earth alone. The pile is lighted by igniting some easily inflammable wood, placed in the chimney formed by the three stakes. When the centre of the pile is well kindled, any sinking-in which has occurred is made good, and the cover extended over the top. Much water now condenses on the inner surface of the cover and round the base, and yellowish-grey smoke escapes from the opening at the base. This is termed the *sweating stage*.

When the colour of the smoke is observed to turn to light-grey this stage is complete, and the covering is at once extended down to the ground, so as to completely envelop the pile ; at the same time the dried and partially charred wood is consolidated by the aid of a pole, the cover being in places rapidly removed for the purpose, and replaced with the greatest expedition possible. The stack is now left to itself for several days, a few small holes, however, being made near the bottom to allow the escape of vapours, and to admit a little air to assist in keeping up the temperature, which would otherwise soon become too low to carry on the operation. In order to complete the charring in the cooler portions of the pile near the surface, it becomes necessary towards the close of this part of the process to make vents at the level of the top of the lowest layer of wood. The thick yellowish smoke which at first issues from these holes gradually becomes blue and thin. They are then stopped, and others opened a little lower down. In large piles several rows of vents may have to be opened one after the other. When it is judged, from the appearance of the smoke issuing from the lowest vents, that the whole pile is completely and properly charred, the cover is made as impervious to air as practicable, after which the stack is again left to itself for several days. Certain precautions have to be observed in withdrawing the charcoal. It is not found expedient to allow the pile to cool down below the point of ignition ; the time required would be considerable, and there would be inevitably a certain amount of loss, besides which it is stated that the charcoal is of better quality when it has been rapidly cooled. The usual method is to uncover a portion of the heap near the bottom, and withdraw the charcoal, rapidly covering up all but the region actually being worked upon. The drawn charcoal is at once quenched with water or covered with damp breeze. The details of this method of charcoal-burning necessarily vary slightly in different localities. The object of the burner, in all cases, is

to conduct the process in such a way as to ensure the combustion proceeding with regularity from the centre towards the circumference, and at the same time from the summit gradually to the bottom. This he does by carefully regulating the direction and amount of the air admitted to the pile, and by making good as they occur any hollow spaces.

In some localities instead of three central stakes one only is used, in which case ignition of the pile is effected by means of a channel left in the base of the pile from the centre to the circumference.



FIG. 2.

Rectangular piles are in favour in some parts of the Continent, more especially in parts of Sweden. The ground should preferably slope about 18 inches in 24 feet. The wood is cut into lengths of about 8 feet, and arranged horizontally so as to form a pile which, when complete, is about 24 feet long, and gradually increases in height from 2 feet to 7 or 9 feet. Air is prevented from having access to the sides of the pile by boards arranged all round, and supported by stakes driven into the ground about 6 inches from the pile and 2 feet apart, the space between the boards and the stack being filled with breeze rammed tightly in. The larger pieces of wood should be placed at the thick end, and, as in the case of the circular pile, all interstices should be filled with small wood.

The top of the pile is covered over with small branches and leaves, and then with a layer of breeze. Ignition is effected from the lower end, where a hole is left for the purpose. This hole is kept open until only light-blue smoke

escapes from the cover in the vicinity, when it is closed, and several others opened in the small end, but somewhat higher up. Following the same indications as before, after a time these holes may be closed also, and the carbonisation should then be complete at the lower end of the pile. In order to carry on the process holes are now made some feet nearer the thick end, and about 2 feet from the ground. In this way the process is conducted until carbonisation has taken place throughout the mass. Each set of holes should be made a little higher from the ground than those preceding. Charcoal can be withdrawn from the thin end when the openings have reached within about 8 feet of the thick part.

A modification of the foregoing method consists in arranging the wood lengthwise, and slanting upwards towards the thick end. In this way carbonisation takes place more rapidly, and the products of the distillation may be collected by means of a pipe inserted at the top of the high end.

All these methods have the advantage that they can be employed close to the spot where the wood is obtained, no special construction being required. In localities in which transport is difficult and expensive—and this is commonly the case where wood in quantity suitable for charcoal-burning is obtainable—the economy of burning on the spot is very considerable, the wood weighing at least five times as much as the charcoal which can be produced from it.

The tar and pyroligneous acid produced during charcoal-burning are collected, in some districts, by means of a specially constructed bed of brickwork or stone, on which the pile rests. Instead of sloping up to the centre the bed is made to slope down to the centre, at which point there is a cavity, loosely covered, and which is connected by a channel with a closed reservoir.

Charring is sometimes conducted in retorts or furnaces of various shapes. This, however, can only be done economically when the wood can be very cheaply transported, or the chief object in view is to obtain pyroligneous acid.

The charring may be effected either by the heat produced by the combustion of a small proportion of the charcoal and products of the distillation, air being admitted in suitable quantity for the purpose, or by heat externally applied, and which may be obtained by burning the volatile products. When charcoal-making is carried on in retorts, the tar is collected by pipes opening into the floor. Care is required to regulate the temperature in retorts, otherwise a too dense charcoal, unsuited for most purposes, is produced.

The yield of charcoal by weight varies considerably according to the way in which the charring has been effected, and also with the age and circumstances attending the growth of the wood. The weight of charcoal produced varies between 15 and 28 per cent. of the weight of the wood employed. Under ordinary circumstances the mean from piles is about 21 per cent. The more slowly the charring is carried on the greater will be the yield.

For commercial purposes charcoal is always estimated by volume, as the varying amount of moisture it contains renders it impossible on a large scale to determine the true weight. The measure employed is a sack.

Charcoal takes up on an average about 12 per cent. of moisture when left exposed to the air. It has also the property of absorbing other gases and vapours, in some cases in great quantity ; *e.g.* one volume of charcoal will absorb about 100 volumes of ammonia vapour, 80 of hydrochloric acid, and 60 of sulphurous acid gas.

PEAT.—The formation of peat is due to the gradual alteration of the substance of plants, especially mosses, under conditions favourable to the presence of much moisture, and consequent partial exclusion of air. There are very great accumulations of peat in different parts of the world, in what are known as peat-bogs. Being a product of recent origin it is always found on or near the surface. As would be expected, the greater the depth from which the peat is obtained the more the original organic structure will have become

obliterated. The structure of peat is favourable to its retaining moisture in considerable quantity, even after it has been removed from the conditions which attended its formation. After what has been said concerning vegetable tissue, it will be readily understood that the organic part of the most recently formed peat has approximately the same ultimate composition as wood. The inorganic or earthy portion of peat is considerably in excess of that of wood. This is traceable to the action of wind and water, carrying matter in suspension, which in greater or less quantity becomes mixed with the decaying plants. The darker the peat is the more will the composition differ from that of the original plants, *i.e.* the more carbon will it contain. It is easy to see that according to its age, and the circumstances attending its formation, peat will vary considerably in its composition and structure. Owing to the way in which peat forms, its ash always contains, in more or less quantity, phosphates and sulphates, if not sulphides.

The specific gravity of peat in the air-dried condition is stated to vary between 0.1 and 1.0 per cent. In order to determine the true specific gravity it would be necessary to expel all the air from the pores. Sir Robert Kane, in writing of the industrial resources of Ireland, says that a cubic yard of light peat, so much used there for domestic fuel, weighs about 500 lbs., good peat, packed close in the form of sods, about 900 lbs., and the densest peat as much as 1,100 lbs.

Frost very much deteriorates fresh peat ; when it is subsequently dried it does not properly consolidate, remaining light and friable.

Irish peat, as it is obtained ordinarily in commerce, contains about one-fourth its weight of water; before drying it contains from 70 to 90 per cent. of moisture. When heated to about 120° C. the organic matter commences to undergo permanent change, and volatile products distil off.

In order to obtain peat in a suitable condition for subsequent treatment it is generally only necessary to drain the

ground by means of trenches, and, having removed the surface material which has not undergone sufficient change, to cut the peat into rectangular pieces by means of a tool consisting of two pieces of iron at right angles to one another. In Hanover it has been found that five men can easily cut and carry to the drying ground 3,000 cubic feet, or 12,000 peats, in a day. After the peat has remained some time on the drying ground, where it has been thrown down in barrowfuls, it is taken away to a suitable position and built up into walls a peat and a half thick. In about a month's time the peats, which by that time are about half dry, are removed, and either stored in magazines or built into large stacks. 1,000 cubic feet of good black peat, freshly cut, yields about 270 cubic feet of thoroughly air-dried peat; the same amount of freshly-cut light-grey peat yields about 700 cubic feet when similarly dried.

In Dublin it has been found that well-dried black peat, which costs 8s. a ton, was less economical than coal for use in locomotives (Percy). On the Continent, notably in the north of France, along the banks of the Somme, the Oise, and the Aisne, and in Prussia, peat-cutting machines are employed. They have the advantage that the peat can be cut without its being necessary to drain the ground, which in itself is often a considerable expense when the peat extends to some depth. The cutter is like an iron box with the bottom removed; it is forced down into the peat, and by means of a lever a flat piece of iron is driven horizontally under it; the cutting of the peat is thus completed, and it can be lifted by the machinery above to any required height, irrespective of the presence of water. The large peats brought up by the cutter are subsequently divided, so as to be more conveniently handled.

Sometimes peat occurs in a more or less muddy condition. This is the case in Holland. Under these circumstances it has to be dredged out by means of canvas bags, usually about eighteen inches deep, fixed to an iron ring.

As much moisture as possible having drained off from the bags, the peat is turned out on to a flat piece of ground fenced round with planks, where it is allowed to dry to a certain extent, after which it is consolidated by men treading over it with boards attached to their feet, or it is beaten down with a suitable flat-headed tool. In this way it is reduced to about half its original thickness, *i.e.* to about six inches ; it is then cut up and air-dried.

In its ordinary air-dried state peat is very tender and crumbles to pieces under pressure ; it cannot, therefore, be employed with advantage in blast-furnaces, neither can it be transported to a distance without much loss. Its great relative bulk as compared with other fuels is also a drawback to its use, not only in respect to cost of carriage, but also on account of the space occupied by it in the furnace. A further objection is the large amount of hygroscopic moisture present.

Amongst the methods which have been employed to render peat more suitable for metallurgical purposes may be enumerated the following :—

- I. Compression of raw peat, with subsequent air-drying.
- II. Compression of air-dried peat, cold.
- III. Compression of air-dried peat, hot.
- IV. Raw peat, $\left\{ \begin{array}{c} \text{consolidated} \\ \text{or} \\ \text{compressed,} \end{array} \right\}$ and $\left\{ \begin{array}{c} \text{air-dried or} \\ \text{artificially} \\ \text{dried,} \end{array} \right\}$ &c.

I. Drying is considerably hastened by subjecting the fresh peat to pressure, so as to squeeze out a portion of the moisture ; in practice much benefit is not, however, found to result when this plan is adopted. Undoubtedly the removal of 20 to 30 per cent. of the moisture enables desiccation to be effected more rapidly, but it is questionable whether there is any real economy in this plan. As the removal of moisture by compression proceeds it becomes more and more difficult and costly, the power required being considerable. The peat obtained is said not to be so satisfactory as that produced by some other methods : it has a tendency to fracture in the

direction of the original bedding. This method, in any case, cannot be applied to muddy peat, as the filtering medium of the compressing apparatus soon becomes choked by the fine particles.

Various machines have been invented for compressing peat before air-drying it. In some, a frame fitted with a flat piston is employed ; in others, various forms of rolls are used. The best filtering material is said to be cloth made of goat's hair.

II. The second method has been successfully applied where good earthy peat could be obtained. The peat having been drained and freed from its superficial covering, is ploughed up into furrows, and then harrowed from time to time until quite air-dried ; it is then forced into moulds by means of stamps weighing about 2 cwts. each. The blocks made in this way have sufficient tenacity to admit of their being cut with a saw. The compression reduces the peat to about two-fifths of its original bulk. A cubic foot of the compressed peat weighs about 80 lbs.

III. In the neighbourhood of Munich and Augsburg the third plan referred to has been adopted. The peat is first air-dried in much the same way as that just described, after which, as it contains many roots, it is passed through a bolting-machine made of wire gauze. The coarse pieces thus separated are used to obtain the necessary heat to raise the temperature of the finer material which has passed through the bolting-machine to 100° C., and to work the compressing machine used to mould the heated peat into blocks. The drying is effected in a series of compartments, heated externally, the peat being gradually transferred from one end of the series to the other by a kind of archimedean screws made of sheet iron. The dimensions of these peat blocks are about 8 × 3 × 1 inches. A single press is stated to yield about 8,000 blocks per hour. Contraction to the extent of one-fifth to one-fourth takes place during the compression ; 10 to 20 per cent. of moisture still remains and there is generally

present about 8 per cent. of ash. The cost of production is stated to be about twelve shillings per ton, exclusive of interest on outlay and wear and tear of plant.

This method certainly possesses advantages, but the expense is considerable, and it is essential that good peat should be used.

It only remains to consider the fourth method.

IV. The peat, if of a muddy nature, is passed through a rotating sieve, having on the inside four radial arms carrying brushes, which traverse over the surface of the sieve, and assist the passage of the fine material through the mesh. The material which passes the sieve is next ground in a pug-mill, somewhat similar to that used by brickmakers, and then pumped into reservoirs formed on flat ground, the sides being built up with clay and boards. In the course of a week, more or less according to the weather, the peat-mud, which at first is about 20 inches thick, begins to crack ; it is then consolidated by men walking over it with boards on their feet, and left to itself for about another week, at the expiration of which it is cut in one direction at intervals of about $3\frac{1}{2}$ inches, and after a further lapse of the same time in the direction at right angles. The blocks are then ready to receive their final drying.

Fibrous peat is either reduced to a pulp by rotating cutters, or is first cut to a certain degree of fineness, and then ground in an apparatus somewhat resembling a coffee-mill.

The peat-mud thus obtained, in some cases, is dried in the way already described, the separation into blocks being effected by means of a frame divided by thin partitions into compartments of the size the blocks are intended to be. The peat is allowed to dry for about a week, by which time it will have acquired a felt-like consistency ; the frame is then pressed down on it so as to indent it. On further desiccation the peat breaks along these lines into brick-shaped pieces, which are then ready for air-drying.

In Weber's method the peat is worked into a pulp, and

is then moulded, without compression, into bricks, which are first air-dried under cover, and then by artificial heat. It is claimed for this peat by some that it is not inferior to compressed peat either in tenacity or compactness. There is a decided advantage in air-drying the peat blocks before subjecting them to artificial heat. Treated in this way they are less likely to crack and get out of shape, and in the end they are denser in structure, there being considerable difficulty in completely drying the centre of the brick without the aid of artificially applied heat. The temperature of the currents of air in the drying sheds ranges between 56° and 63° C.; the heating is effected by flues extending from one end of the sheds to the other, and arrangement is made for the admission of fresh air, and for the escape of that charged with moisture. The drying is complete in from eight to twelve days. The air-dried peat, when first introduced into the drying-house, softens and swells until it has become of the same temperature as the surrounding atmosphere, when it rapidly shrinks, and becomes firm and dense. The weight of a cubic foot of peat prepared in this manner is said to be about 34 lbs. It still contains about twelve per cent. of moisture.

In some other methods the general details are much the same as in those already described; but the blocks are moulded by pressure, by means generally not differing greatly from those employed in making ordinary bricks.

Various other methods, in principle the same as Weber's, but differing in detail, have been employed. In some of these, kilns are so arranged that the hot waste gases from furnaces can be passed through them, either from the top downwards, or in the reverse direction. The temperature of the gases as they enter is regulated, by the suitable admission of air, so as not to exceed 150° C. in contact with moist peat, and 120° with dry peat, and not to fall below 100° . Instead of waste gases, air specially heated is substituted in cases where the former are not available for the purpose. The maximum rate of flow of the gases or air on entering is

about 400 cubic feet per minute, for a kiln which has about 4,000 cubic feet of drying space. In different arrangements this will necessarily vary somewhat. In the best forms of apparatus the working is continuous, peat being charged in at the top as fast as it is drawn at the bottom.

One of the most important points to keep in view in constructing a kiln is, that the hot air should be made to enter in such a way as to come first in contact with the peat which is in the moistest state. In this way there is a greater economy of heat, and less danger of the peat igniting. It is also important that the walls of the kiln should be as air-tight as possible, otherwise local cooling of the peat would take place, and the desiccation would proceed unequally.

Machines have been invented for removing the moisture from peat by centrifugal action ; they have not as yet had much success. The chief difficulty in this method is to prevent the sieves from becoming choked up, and it is not easy to see how this can be overcome.

There is an important objection to peat made into bricks in the ordinary way, namely, that when charged into a furnace they are very apt to fall with their flat faces together, leaving insufficient space for the passage of air. The best remedy would be to make the peat into ball-shaped pieces ; but this is not practicable owing to the expense involved. In the pulping process the peat can be forced out of the machine in a continuous cylindrical piece, and then cut up into lengths. These cylindrical pieces are better suited for burning than the square bricks.

In consequence of the large amount of ash in peat from low lands, and the nature of that ash, it is not well suited for use in the metallurgy of iron, in operations in which the fuel and the metal are contained in the same compartment. Peat can, however, be satisfactorily employed when the combustion is carried on in a separate chamber, which is every day becoming more generally the case, the only important exception being the blast-furnace in which the ore

is smelted for pig-iron. Prepared peat can also be employed in the blast-furnace, when a high-grade iron is not required. The useful calorific value of peat is slightly lower than that of wood, in the condition in which both are usually obtained in commerce.

Peat can be made into charcoal, but the product is not by far so satisfactory as in the case of wood. In parts of Germany it is used to a certain extent, in admixture with wood charcoal, in high-blast furnaces, and common forge-fires.

LIGNITE AND COAL.—It has been stated that peat is of recent origin. But if these things are going on under our eyes at the present day, it is reasonable to expect that something similar has occurred in past geological ages. During those remote times, the extent of which, as evidenced by geological science, is so vast as to be almost beyond the grasp of the average human mind, there flourished a luxuriant growth of plants and trees, extending over large areas. It is to those plants and trees that we owe the wealth of coal which has given such an enormous impetus to civilisation during the last hundred years. Vegetation is the natural store-house of the heat of the sun which reaches the earth. Under its influence the germs of vegetable life assimilate carbon, hydrogen, and oxygen from the moisture and carbonic acid which form part of our normal atmosphere. The compounds thus formed, when raised to a sufficiently high temperature in the presence of air, yield up the heat absorbed during their formation. If complete combustion be effected, not a fraction of the heat which originally contributed to the growth of the plant is lost. The vegetable substances were built up from carbonic acid and moisture, and these bodies are the sole products of the *complete* combustion of organic matter of every kind.

It is perfectly well established that lignite and coal, equally with peat, are not formed from vegetable matter which has undergone *decay* from exposure to the atmo-

sphere. It follows that the vegetation from which coal has originated must have been protected from direct atmospheric influences, and there is no reason to doubt that this took place in a similar manner to that which accompanies the formation of peat, *i.e.* that the coal-plants grew in swampy ground, probably forming the estuary of some large river. Owing to subsidence (until the earth's crust in that region was below water) the coal plants ceased for a time to grow, and the deposit of lifeless vegetable matter became covered with a layer of earthy material, borne down by running water and gradually deposited. After this sedimentary deposit had increased to a certain thickness, vegetable life again became possible. In this way a series of strata have been built up, in which seams of coal alternate with layers of clay, limestone, sandstone, and ironstone. These deposits, which are found over areas hundreds of miles in extent, are called the *coal-measures*. The nature of the coal, as we now find it, will depend on the time which has elapsed since its formation, and the amount of pressure to which the strata have been subjected. The longer the time and the greater the pressure, the more hydrogen and oxygen will have been eliminated, and the more closely will the residual substance approximate to pure carbon. Although these changes are so gradual that no alteration in any seam of coal would be apparent to a careful observer during a human lifetime, yet that they go on is conclusively shown by the accumulation of the resultant gases in mines where sufficient ventilation to carry them off is not supplied.

The following classification of coal is due to Dr. Percy. It will be observed that it is based on variation in the chemical composition of the substance :—

Lignite.

Bituminous Coal { Non-caking, rich in oxygen.
Caking.
Non-caking, rich in carbon.

Anthracite.

M. Grüner's classification, given below, has in view the industrial application of each particular kind. It differs from Dr. Percy's only in being more detailed.

Grüner's Classification.

Names of the five types or classes	Composition per cent. of the organic constituents			Number of parts by weight of oxygen,* taking the weight of hydrogen as unity	Percentage of coke	Nature and appearance of the coke
	Carbon	Hydrogen	Oxygen*			
Dry coals, burning with a long flame . }	75-80	4.5-5.5	15-19.5	3-4	50-60	{ Pulverulent, or, at the most, fritted
Fat coals, burning with a long flame, or gas coals . }	80-85	5-5.8	10-14.2	2-3	60-68	{ Caked, but very friable
Fat coals, properly so-called, or furnace coals }	84-89	5-5.5	5.5-11	1-2	68-74	{ Caked, moderately compact
Fat coals, burning with a short flame, or coking coals . . . }	88-91	4.5-5.5	5.5-6.5	1	74-82	{ Caked, very compact, but little friable
Lean (maigres) coals, or anthracites . . }	90-93	4-4.5	3-5.5	1	82-90	{ Fritted or pulverulent

* Including nitrogen, the proportion of which is stated rarely to exceed one per cent. of the organic constituents.

The reader must carefully bear in mind that whatever classification is adopted it is only for convenience of descrip-

tion. As has been shown by Dr. Percy in the following table, there are coals the analyses of which form a series unbroken in continuity from wood to anthracite. Peat appears to be inserted in the table not because it properly forms a link in the chain, but rather because it is a product of more recent origin than lignite, and therefore approaches more in composition to ligneous tissue, thus filling the gap which time has made between wood and lignite. For sake of comparison carbon has been taken in all cases as 100 :—

Table showing the gradual Change in Composition: from Wood to Anthracite.

Substance	Carbon	Hydrogen	Oxygen	Disposable hydrogen *
1. Wood (the mean of several analyses)	100	12·18	83·07	1·80
2. Peat (the mean of several analyses)	100	9·85	55·67	2·89
3. Lignite (mean of 15 varieties).	100	8·37	42·42	3·07
4. Ten-yard coal of the South Staffordshire basin .	100	6·12	21·23	3·47
5. Steam coal from the Tyne	100	5·91	18·32	3·62
6. Pentrefelin coal of South Wales	100	4·75	5·28	4·09
7. Anthracite from Pennsylvania, U.S.	100	2·84	1·74	2·63

* The term *disposable* hydrogen is here used to signify the amount in excess of that required to form water with the oxygen present.

Lignite.—This term is synonymous with the *braunkohle* (brown coal) of the Germans. It is employed to designate substances which form the link between peat on the one hand and true coal on the other. In those kinds which approximate most closely in their composition to peat the only essential difference is that they have been derived from woody matter, whereas peat owes its origin to mosses, for the most

The products of distillation of dry lignite average :—

Carbon	40-50
Water	20-15
Tarry matter	16-14
Gases	24-21
							<hr/> 100

Lignites, with few exceptions, are *non-caking*; i.e. when powdered and strongly heated, with exclusion of air, the particles will not cohere together and form what is known as *coke*.

Bituminous Coal.—The kind of coal into which lignite in its most altered state merges is bituminous coal. Although these two substances in some cases may be difficult to distinguish one from the other, yet that is far from being the case when average specimens of either are considered. The density of bituminous coal is, as a rule, greater than that of lignite or brown coal; it contains less water in its natural state than lignite, and after drying at 100° C. is also less hygroscopic. It has previously been stated that bituminous coals may broadly be divided into two classes, viz. caking and non-caking. It is within the experience of all of us that some house-coal soon after it has been thrown on the fire begins to soften, finally becoming more or less pasty in consistency, and giving off gas which burns with a bright flame. If a lump of such coal be heated, with partial exclusion of air, then it will be found that after a short time gas ceases to come off, and the lump, which on the first application of heat became pasty, will now be a more or less hard coherent mass of coke. Coal which possesses this property is technically described as a caking coal. There is every degree of caking, some coals caking very slightly, whilst others may almost be said to fuse when heated. A non-caking coal, when treated in the way just described, leaves a residue the particles of which do not cohere in a marked degree.

Before leaving this part of the subject it will be profitable to glance briefly, but systematically, at the different varieties of bituminous coal, commencing with those most nearly

allied to lignites. For this purpose the French classification framed by M. Grüner can be advantageously followed, substituting, however, for the terms *dry*, *fat*, and *lean* the more familiar expressions caking and non-caking.

1. *Non-caking Coal, with a long flame.*—These coals when treated for coke crack, but preserve their original form ; and if originally in powder the particles do not become consolidated. Coals of this kind are hard, compact, and but little friable. A cubic metre of them weighs 700 kilogrammes. The fracture is often conchoidal, and of such a nature as to have given rise to the name *splint-coal*. It is not common in France or Germany, but in Derbyshire, Staffordshire, and Scotland it exists in quantity ; in colour it is rarely perfectly black ; the powder is brown. The calorific power ranges between 8,000 and 8,500, taking the two extremes of the class and assuming the absence of moisture and extraneous matter. One part of the dried coal will convert 6 to 6.5 parts of water at 0° into vapour at 112° C., when only just sufficient air for combustion is supplied. In the lower part of the basins the coal passes into that comprehended in the next class, viz. :—

2. *Caking Coals, with long flame.*—As the title indicates, these coals coke ; those nearest in nature to class 1, however, when powdered barely become cemented together on being heated. They go by the name of cherry-coal in the North of England. In hardness they do not equal class 1, and the fracture tends less to be conchoidal ; they are also blacker and brighter. A cubic metre weighs from 700 to 750 kilogrammes. Like the last described, they burn with abundance of flame and smoke ; they take fire readily and burn rapidly. The coke is light, friable, and porous, and therefore unsuited for metallurgical purposes ; but for the manufacture of lighting-gas these coals are very valuable, the gas, though in quantity not so great as that from the non-caking coal of class 1, being of greater illuminating power. The coke is, besides, good enough for use where it will not be

subjected to much pressure or a strong blast. The calorific value ranges between 8,500 and 8,800. According to trials made at Portsmouth and Woolwich, 1 part of this coal in its ordinary condition (say containing 5 to 8 per cent. of water and ash) will convert 7 to 7.75 parts of water into vapour. Coal of this description occurs notably in the neighbourhood of Newcastle and in South Wales. To the north of Newcastle it is mostly steam-coal, and to the south gas-coal ; it is, however, all one basin.

3. *Caking Coal, properly so called.*—Coals of this type have a less lamellated structure than the last ; they are also blacker, brighter, and much softer on the average, and the flame is shorter and more brilliant, giving less smoke. When heated they soften, even to fusion in some cases. Although they contain less volatile matter than those last considered, they swell up more in coking. The properties of this coal render it well suited for use in blacksmiths' forges and for making coke ; though for the latter purpose the coals of the next class are even better suited. The weight of a cubic metre is 750 to 800 kilogrammes, and the calorific power 8,800 to 9,300. In the condition in which they are obtained in commerce, containing from 5 to 15 per cent. extraneous matter, including moisture, 1 part of these coals will vaporise 7.5 to 8.5 of water. This description of coal is found largely on the Continent : in France, particularly in the St. Etienne basin and the north ; in Belgium, in abundance in the neighbourhood of Liège and in the Mons basin ; also in Westphalia. In England they occur more especially in Durham and Yorkshire.

4. *Caking Coals, with short flame.*—These coals have often the appearance of being made up of alternate bright and dull layers. The character of their fracture resembles class 3 ; they are less rich in volatile matter, and generally very friable, *i.e.* crumble under moderate pressure. They inflame with difficulty, burning with a short, blueish-white and generally smokeless flame. This coal is the best suited

of all for coking, the product being compact and hard, and relatively large in quantity. A cubic metre weighs about 800 kilogrammes. This class possesses the highest calorific power of all, viz. 9,300 to 9,600.

In England this coal is located more especially in the neighbourhood of Cardiff, in South Wales ; in France, near Creusot, and in the St. Etienne and other basins to some extent. The trials made of the Welsh coal of this type show that it is capable of vaporising 9 to 9·5 parts of water per unit of coal. As compared with the caking coal from the North of England basin, they have a greater actual calorific value, but the northern coal burns more rapidly, and in consequence produces a greater amount of heat in a given time.

5. *Non-caking or Anthracitic Coals, and Anthracite.*—

The first of these are characterised by dull streaks ; they are harder than class 4, and become more and more so as they approach anthracite in nature. The weight of a cubic metre is about 850 kilogrammes. They give but little flame, and often decrepitate in the fire, an extremely inconvenient property, for if they split up into small pieces it becomes difficult to keep up a sufficient draught through the fire to maintain the requisite temperature. Some kinds are, however, successfully employed in the blast-furnaces of South Wales ; they are found in the neighbourhood of Swansea and Merthyr Tydvil. They also abound in Pennsylvania, U.S., but are not of frequent occurrence on the Continent. Their calorific power ranges between 9,200 and 9,500, and their steam-raising power is about 8·5 to 9·5 when, say, 6 per cent. of ash, &c. is present. It is difficult to ensure anything like complete combustion.

With *anthracite* we have reached the end of the series. In the most highly altered kinds it is brilliantly black, and even when in powder has not a brown shade in it. It is hard, brittle, and generally conchoidal in fracture. It does not soil the fingers. The weight of a cubic metre is usually

850 to 900 kilogrammes, but it may be even higher if much ash is present. When heated it often decrepitates—even when gradually heated. American anthracite is said not to have this property. Anthracite burns with difficulty, requiring a strong draught. The calorific value is much the same as that of the anthracitic coals, but to obtain the same effect greater care is required. South Wales, Pennsylvania, and the French Alpine districts are the localities in which anthracite is most abundant.

General considerations.—It has already been pointed out that coals having the same elementary composition may have a different calorific power, owing to the different way in which the elements are combined in the two cases. It is for a similar reason that overlapping takes place in the classification just considered. An analysis will tell us approximately in which class a certain coal should be placed ; it is, however, only on submitting the fuel to distillation that the point can be conclusively settled. One coal may contain a greater percentage of carbon than another and yet produce less coke. It is important to note that these considerations apply generally only to coals from different basins. Coals from the same basin have, with few exceptions, been subjected to like conditions, and although they may be in various stages of alteration, yet the series is perfectly continuous and free from overlapping.

It will be observed, on referring to the table, that at the same time that the percentage of carbon gradually increases the disposable hydrogen also increases ; *i.e.* the oxygen is eliminated in a greater degree than the hydrogen, until the fourth class is reached, when the hydrogen begins to decrease down to the most altered form of anthracite. This explains the fall at the same point in the calorific power, already referred to.

In selecting a coal it is important to ascertain the extent to which it cokes. If it cokes very much the grate may become blocked. In order to obviate this, in some cases it

is necessary to mix a certain proportion of non-caking or *free-burning* coal, as it is called, with the *binding* coal.

Ash of Coal.—For practical purposes it is very important to note, not only the quantity of ash, but also its nature. This is done by burning off the combustible matter in a platinum crucible. In examining the ash the first thing to consider is the amount of oxide of iron present ; this is indicated by the colour. The oxide of iron in the ash is mainly derived from iron pyrites, it will therefore roughly indicate the amount of sulphur in the coal. At the temperature of most furnace operations oxide of iron forms a fusible compound, termed *clinker*, with the remainder of the ash. This clinker tends to block the grate and prevent proper access of air ; its consistency is determined by the proportion of oxide of iron. As a rule, the less clinker a coal makes the better. In one particular case, however, it has been made to serve a useful purpose, *viz.* in South Wales, where a bed of clinker, supported by a few bars, is commonly allowed to form to a depth of from 12 to 20 inches at the bottom of the furnace. This clinker bed is used as a substitute for a grate, being broken up from time to time by means of a bar, so as to let sufficient air through. By this contrivance inferior small coal can be burnt, which would be useless in an ordinary grate.

Sulphur occurs in coal, probably, in three conditions, *viz.* as iron pyrites, in organic combination, and as sulphate. It is, however, mainly present as iron pyrites. Coal containing much sulphur is not suited for use in those processes in the manufacture of iron in which the metal comes in direct contact with the fuel ; neither is it suited for the manufacture of coke. There is not the same objection to the use of sulphurous coal in reverberatory furnaces.

Phosphorus, in more or less quantity, nearly always occurs in the ash of coal. This should not be lost sight of in the selection of coal for certain processes in the manufacture of iron and steel.

Arsenic sometimes exists in coal, being doubtless present as arsenical iron pyrites. *Antimony*, *lead*, and *copper* have been occasionally noticed.

Weathering.—Coal when exposed to the action of the atmosphere undergoes certain changes, due to the action of oxygen in the presence of moisture. A certain amount of carbon and hydrogen is eliminated in combination with oxygen, and at the same time some oxygen becomes fixed ; as shown by the fact that, if a coal is dried at a temperature not exceeding 100° C., it will lose weight up to a certain point, owing to the removal of moisture and gases, and will then begin to gain in weight, the oxygen taken up more than compensating for the carbon and hydrogen lost.

The caking property of coal appears in some cases to be materially affected by weathering. It is said that a coal obtained at Penclawdd, near Swansea, even in so short a space of time as two days loses its power to cake. Many coals on exposure for a few months become much deteriorated in quality. It has not yet been determined whether the change is due to the escape of volatile matter or to oxidation ; it is probable that both conduce towards the result, oxidation, however, being the initial cause of the mischief.

COKE bears the same relation to coal that charcoal does to wood. The first production of coke appears to have been due to an endeavour to find a substitute for wood charcoal, coal having been found not to answer the purpose. Indeed, the only method for the making of coke practised for a long time was that of burning coal in piles, much in the same way as was done with wood for the production of charcoal. The loss in this method is very considerable. As in the case of wood, coal is only treated in retorts when the principal object in view is to obtain the volatile products, as in the manufacture of illuminating gas. The coke produced in this way is never so good as that obtained by other methods, to be referred to ; it is much more tender and friable and not well suited for

use where considerable pressure will be put upon it, as in the blast-furnace or cupola.

In coking in piles, owing to the greater density of coal, the same precautions are not required as in charcoal-burning. The process is, however, when conducted to the best advantage, essentially the same. The central chimney is made in brickwork, air spaces being left all the way up, and a cast-iron damper fitted to the top. The cover of the pile consists of powdered coke or rubbish, there not being the same occasion for a yielding cover as in charcoal-burning. When highly bituminous coal is being coked, it is often desirable to allow the heap to become thoroughly ignited throughout before applying the cover ; in such cases, also, owing to the tendency of the coal to fuse, great care has to be taken to keep the air-channels properly open. Some prefer to build the pile on moist ground, with the intention to promote the removal of sulphur ; this point will be discussed later on. The presence of moisture is also considered advantageous in checking too rapid combustion. It must, however, not be lost sight of that steam is decomposed by carbonaceous matter with the production of carburetted hydrogen, hydrogen, carbonic oxide, and carbonic acid. The loss incurred in this way must, therefore, be set against the advantages which may accrue.

In some localities rectangular kilns are in vogue, and very excellent coke is said to be produced in them, with less loss than occurs in the ordinary piles. These kilns consist of two parallel walls of brickwork, about 5 feet high, 8 feet apart, and 40 to 60 feet long ; the floor and the inner portion of the walls are of fire-brick. In each wall is a series of openings (E F), about 2 feet apart, and leading from them are vertical channels (G H). In order to charge the kiln, one end is first bricked up ; coal slack is spread over the bottom, damped, and stamped down until it reaches up to the openings (E F), *i.e.* about two feet from the bottom. Slightly tapering pieces of wood, about 6 inches in diameter at the

thick end, and long enough to reach across the kiln, are placed with their ends in the openings (EF). The remainder of the kiln is then completely filled with slack, which is damped and stamped down as it is charged in ; the top is then covered with coal dust or loam, the end through which the charging has been effected having first been bricked up. The pieces of wood are now carefully drawn out ; thus channels are formed through which ignition can be effected, and which also serve as means by which to carry on and

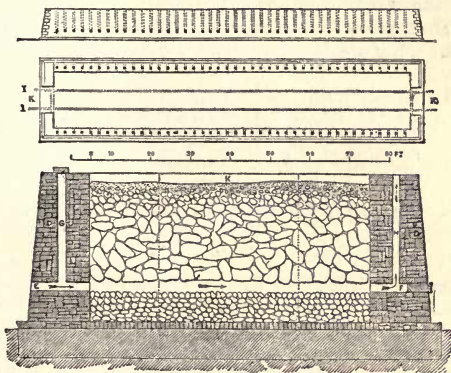


FIG. 3.

regulate the process, by the admission of a suitable quantity of air. Before lighting the kiln, by means of easily inflammable sticks, all the vertical flues or chimneys on one side are closed, and also the horizontal openings on the opposite side. When the ignition has travelled through to the opposite side, the channels in the brickwork which were open are closed, and the others opened. This reversal of direction takes place, at the discretion of the workmen, every two or three hours until the process is complete ; the direction of the

wind at the time will have an important influence on the working, if not carefully counteracted. This method requires about eight days for completion. Where wider kilns, *e.g.* 14 to 15 feet, are used, the channels are constructed by means of lumps of coal instead of the wooden poles. Unless great care be exercised very considerable loss may occur through the burning away of the coal. There is no doubt that both this plan and that previously described are very wasteful as compared with the methods of coking in ovens, which are now almost universally adopted.

The simplest form of oven is that known as the bee-hive oven. They are sometimes circular, but often square or

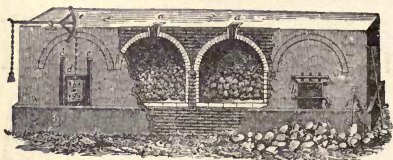


FIG. 4.

oblong. The height and diameter vary up to about ten feet. These ovens are usually built in blocks, to economise heat ; the dividing walls are about two feet thick, the inner facing being of fire-brick. At the top is an opening for the escape of the volatile products ; through it the ovens are usually charged, from coal-trucks brought immediately over the ovens by means of a branch line of rails. In front is a door through which the coke is removed ; charging is also sometimes effected through this opening. The door often consists of a perforated plate of iron ; more generally it is formed of fire-brick enclosed in an iron frame, attached to a lever. Through the door air is admitted in suitable quantity, some of the spaces being blocked with clay when it is necessary to check too rapid action, which usually first takes place after the expiration of about two or three hours

from the commencement. After the lapse of about twenty-four hours the door is entirely plastered up, and in another twelve hours, when flame will no longer issue from the top opening, that also is closed. The oven remains undisturbed for another twelve hours ; the door is then opened, and the coke raked out into iron barrows or trucks, the ovens being built at a convenient height from the ground for the purpose. The tool employed to withdraw the coke is termed a *drag*. It consists of a piece of flat iron, having attached to it at right angles a rod long enough to project from the front of the oven when the flat piece is in position at the back. By means of a windlass the drag is pulled out, and with it the whole of the coke. In some places the coke is pushed out from the back. It is important, in any case, that the oven should be rapidly cleared, so as to avoid its being unduly cooled down. It is preferable to quench the coke before drawing it, failing which it should be done immediately afterwards.

In principle these ovens are simply reverberatory gas furnaces. The heat necessary to carry on the dry distillation of the coal is obtained from the combustion of the volatile matter by means of air admitted *above* the fuel ; the heat thus produced is radiated from the top mainly, but also from the hot sides and bottom, and maintains the required temperature. The coking gradually extends from above downwards, and from the bottom and sides towards the middle. If the process is properly conducted only a slight consumption of the coke takes place, since the air should only come in contact with the gaseous compounds above the coke. Much economy is effected by leading the excess gases under boilers, and then burning them. More than enough steam for all the purposes of a colliery can be raised in this way.

The bee-hive ovens are certainly not so economical as some which have since been invented, yet, owing to their relative small cost, the ease with which they may be put up,

and the good quality of coke producible in them, they have up to the present very successfully held their own. It is observable, however, that gradually some more modern and economical inventions are superseding the old bee-hive oven. It is natural that progress in these matters should be slow : a manufacturer rightly hesitates before giving up an old tried servant for another who claims to have greater accomplishments, but whose trustworthiness has yet to stand the test of time.

In order to put ourselves in a position to fully appreciate the claims of modern inventions of this kind, let us consider, *seriatim*, what are the main objects which it is desirable to attain.

I. The introduction of air in such a way as to completely burn the gaseous products of distillation, but not the coke.

II. The prevention of loss of heat by radiation and conduction, and during the drawing of the coke.

III. The promotion of uniform coking, from the exterior in all directions towards the centre.

IV. Rapid coking. The quality is improved and the yield increased by rapid coking.

For purposes of illustration, the Appolt, Coppée, Carvés, and Pernolet ovens have been selected.

The *Appolt* ovens are largely used abroad, and coals differing very considerably in their character are treated successfully in them.

They consist essentially of a large brick chamber, containing, in the most recent erections, eighteen compartments or retorts (A), tied together at a sufficient number of points to ensure solidity, but having a continuous free space round all of them. Charging is effected through an opening at the top of each retort ; the bottom is provided with a cast-iron door, over which a layer of coke dust is spread before charging. The retorts are about 4 feet long and 1 foot 6 inches wide at the base ; they taper slightly, so that at the top they are about 3 feet 8 inches by 13 inches ; in height they are

about 13 feet. At about 18 inches to 2 feet above the bottom of each retort are two rows of small horizontal openings (c), say 5 inches by 2 inches ; it is through these openings that the volatile products of distillation pass into the external spaces, and are there burnt, by means of a suitable quantity of air admitted through openings in the long sides of the oven. In treating highly-caking coal, other vents further up are required. The products of the combustion

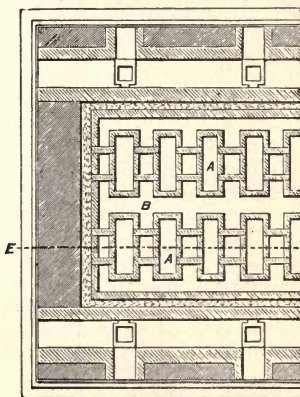


FIG. 5.

of the gases are taken off by means of sixteen flues, four at the bottom and four at the top of each long side of the chamber. Each set of four is led into a longitudinal flue, divided into two by a partition, and connected with a chimney similarly partitioned. Dampers are placed in the vertical flues, where they are connected with the horizontal flues. In this way the heat throughout the oven can be regulated as required. The distance between two compartments is about 8 inches to 10 inches. Between the walls

forming the chamber and the outer mass of brickwork, which is necessary to avoid, as far as practicable, loss of heat by radiation and conduction, there is a layer of loose non-conducting material, such as sand or brick-earth, which allows the brickwork to expand and contract, and thus prevents the oven pulling itself to pieces, and at the same time assists in keeping the heat in. In removing the coke it is allowed

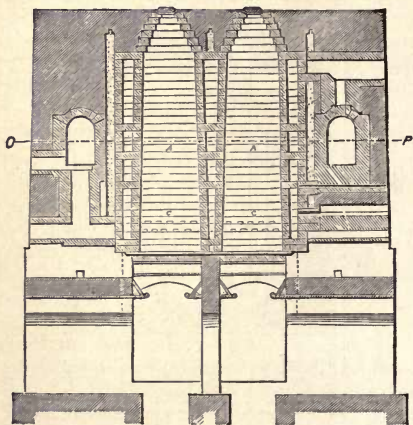


FIG. 6

to fall on sloping iron plates, to avoid its being unduly broken.

It is desirable that the coals should contain at least 20 per cent. of volatile matter, but, on the other hand, they should not be too strongly caking, for then the coke is very troublesome to remove from the oven. Any difficulty of this kind is easily met by mixing coals of different natures. The ovens are charged in succession ; the eighteen compart-

ments will take about 24 tons of coal ; the coking is complete in about twenty-four hours. Each retort is charged again immediately after the withdrawal of the coke, so that the process once in working, there is always heat sufficient to start the coking of each fresh charge.

Moist washed coal can be used in these ovens without inconvenience.

The waste of coke by oxidation is in this oven reduced to a minimum, as the air, even should there be cracks in the

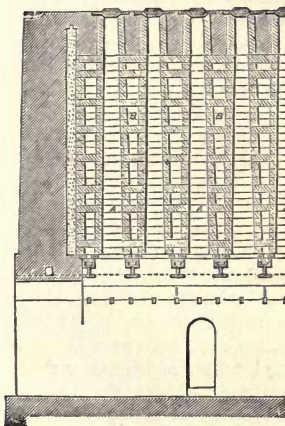


FIG. 7.

sides of the oven, cannot reach the inside of the retorts, the gases in the surrounding space forming an absolute barrier. In this respect then, the Appolt oven has a marked advantage over the ordinary ovens, in which all the air is admitted into the coking-chamber itself. Another advantage is the very large extent of heating surface, and the comparatively

small size of the retorts, whereby rapid carbonisation is ensured, and thereby proportionate density in the coke. The combustion of the gases is very perfect, the arrangement of the oven facilitating their thorough admixture with the proper amount of air. The arrangement for withdrawing the coke enables it to be done very rapidly, and the cooling of the ovens is proportionately less. The height of the retorts acts beneficially, as it tends to increase the density of the coke. The construction of these ovens is also favourable to the production of coke uniformly from the sides of the retort to its centre. Owing to the bad conductivity of coke for heat, the process goes on more and more slowly as the thickness of the layer inside the retort increases.

The addition of water to caking coal is said to act beneficially in preventing it from swelling up too much and becoming fixed tightly in the retort. The yield of these ovens is said to be as great as that which can be obtained in a crucible on the small scale, *i.e.* 10 or 12 per cent. more than that obtained in ordinary ovens.

Coppée Ovens.—The principle of these ovens is essentially that of the Appolt system ; the method of carrying it out is, however, somewhat different. In the Coppée system the retorts are long horizontal chambers, very much the same thing, in fact, as if the retorts of the Appolt oven were placed horizontally instead of vertically. The retorts (A), are about 9 metres long, 480 millimetres at the back, 430 at the front, and about a metre high at the crown of the arch. The tapering from the back to the front is to facilitate the removal of the coke, which is effected from the back by means of a ram. At each end of a retort there are two doors, the lower one being about 3 feet in height, the upper about 1 foot. In the partition wall between each two coking-chambers there is a series of twenty-eight vertical flues, which lead the volatile products from the top of the retorts to a horizontal flue (C), passing under *one* of each pair of retorts in the direction of its length. Smaller vertical flues, through which air is

admitted to effect combustion, communicate with the top of each of the flues leading from the coking-chambers. The products of combustion having passed under one retort, as just described, are led into a similar channel (D) under the other retort, from the front end of which they are drawn off into the main flue (E), leading to the stack. Beneath the horizontal flues, under the retorts, are a series of channels,

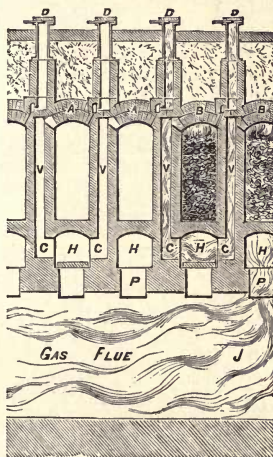


FIG. 8.

through which cool air is circulated by means of a separate stack, the object being to keep the foundations from being damaged by excessive heat. Besides the air-channels already described for effecting combustion in the vertical flues, there are others which admit a certain amount of air into the top of the retorts. The quantity of air admitted is regulated by dampers. Having to pass for some distance

through the hot masonry, the air supplied for combustion becomes heated before mixing with the volatile products

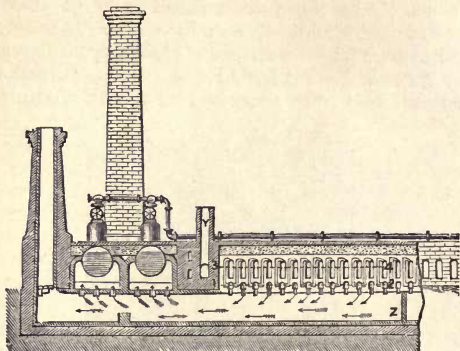


FIG. 9.

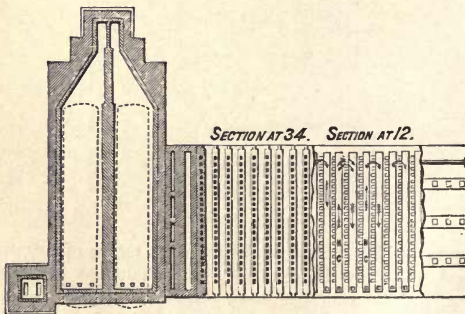


FIG. 10.

from the coal. The products of combustion, before being taken up the stack, are often first led under boilers, and their

heat thus utilised to the utmost. The top of the ovens is covered with a thick layer of rubbish, to keep in the heat as much as possible.

It will be observed that the way in which the air is admitted to the ovens partly resembles the method in the bee-hive oven, and partly that of the Appolt. The amount of air, however, which is admitted directly into the coking-chamber of the Coppée oven is very small, and cannot burn the coke if properly regulated.

These ovens are particularly suitable for the treatment of coals which are not very bituminous, and therefore difficult to coke in ordinary ovens. The coal must be crushed to about the consistency of very coarse meal ; it is therefore easy to wash it before coking, if desired. It is found most convenient to build the ovens in batches of thirty, and to work them in pairs ; the charging is so arranged that one oven of each pair receives a fresh charge when that in the other is half coked.

Coke produced in ovens is much harder and denser than that made in open heaps, and called *yard-coke*. The latter kind of coke, and that produced in retorts when coal is distilled for the sake of its volatile products, may be used with advantage with cold blast, when the pressure of superincumbent material is not too great.

It has been pointed out that a coal which is highly caking may give trouble in an Appolt or a Coppée oven by rendering it difficult to withdraw the coke, and that this difficulty can be overcome by mixing with the binding coal a certain amount of free-burning coal. The principle herein involved may be further extended. The *slack*, or small of non-caking coal, may be utilised for the production of coke by mixing it with bituminous coal. The two kinds of coal should be ground separately, the caking being reduced to a fine state of division, whilst the non-caking may be in pieces about the size of a pea. In this way even anthracite may be used, the particles becoming firmly cemented together by the

caking coal. In this case it is necessary to add a small amount of pitch as well.

Other similar means have been resorted to for the utilisation of small coal, in which various carbonaceous substances are employed to act as a cement. Thus, anthracite or coke breeze mixed with tar will produce a solid coke on being subjected to heat. *Patent fuels* consist of mixtures of this kind, moulded into the form of bricks with or without the application of heat. Resin is said not to give satisfactory results as a cementing material unless mixed with tar. Mucilaginous substances prepared from farinaceous bodies are also used. The use of patent fuels is almost entirely confined to raising steam for navigation and on railways, for which purposes they answer very well, as the space occupied is small compared to that required for ordinary fuel. To render the blocks impervious to moisture, they are sometimes dipped in coal-tar oil immediately on being withdrawn from the drying oven.

Removal of Sulphur from Coke.—The iron pyrites (FeS_2), which exists in more or less quantity in every coal, undergoes decomposition during the coking process ; one half of the sulphur is distilled off, and unites with oxygen to form sulphurous acid (SO_2), during the combustion of the volatile products. The remainder of the sulphur will be found in the coke, in combination with the iron, as FeS . The problem we have before us is how to remove this residual sulphur from the coke, supposing it to be present in such quantity as to detract from the value of the coke for furnace purposes.

It is a matter of common observation, that if water be thrown on red-hot coke or coal a strong odour of an objectionable kind is at once perceptible. This arises from the decomposition of the protosulphide of iron, oxide of iron being formed, and the offensive gas, sulphuretted hydrogen (SH_2). From this it might be argued, that in order to desulphurize coke it would only be necessary to pass steam into the oven. It must, however, be borne in mind that the

amount of sulphur in coke, as compared with the carbon, is very small, and that it is practically impossible to ensure the steam permeating the coke throughout. The impossibility of removing anything like the whole of the sulphur in this way is still more apparent when it is remembered that red-hot carbon decomposes steam, with formation of carbonic oxide, carbonic acid, marsh gas, and some free hydrogen. It is easy to overrate the amount of sulphur removed by steam if it be judged only by the sense of smell, the odour of a little sulphuretted hydrogen being very powerful. Desulphurisation by means of air, either at the ordinary pressure of the atmosphere or at higher pressures, has been tried. It is obvious that the objections to the use of steam apply equally to air.

The addition of such substances as carbonate of soda, lime, carbonate of lime, oxide of manganese, chloride of sodium, &c., has been tried, but only with indifferent success, not warranting the extra expense entailed. This method has for its object the transference of the sulphur from the iron to the substance added, which once having entered into combination with the sulphur will, for instance, prevent its passing into iron melted with the coke in a cupola.

It follows, from the foregoing considerations, that at present there is no satisfactory method known by which sulphur can be removed from coke. The most satisfactory way of meeting the difficulty is to disintegrate the coal before coking it, and subject it to a washing process by means of which the shale and pyrites which it contains can be, by reason of their greater specific gravity, to a considerable extent removed. This treatment cannot be applied to non-caking coals unless they are subsequently mixed with a sufficient quantity of caking coal to make them coke.

Collection of the Products of Distillation of Coal.—This is done, as has already been stated, in the manufacture of ordinary lighting gas; but the coke produced, when the chief object is to obtain as great a quantity of volatile products

as possible, is lighter and more porous than that made in ovens such as have just been considered. It has been attempted to produce dense hard coke, and at the same time to collect the tar, oils, ammonia, and gases. The ovens may be heated entirely by refuse coke or by the gases, or partially by gases and partially by coal or refuse coke.

The *Pernolet* oven is one of the most important of this class. It was patented in 1862; an almost exactly similar patent was taken out in 1850 by Pauwels and Dubochet. Pernolet claims, firstly, to make coke suitable for metallurgical, railway, and other purposes, in greater proportion than when manufactured in the ordinary manner; secondly, gas suitable for burning and heating; thirdly, tar, and different oils obtainable therefrom; and, fourthly, ammonia and ammoniacal salts.

The 'coking is to be conducted 'very slowly;' air is excluded as far as possible after thorough ignition has been effected; the products of distillation, which are either drawn off by the draught of a high chimney or by exhausting-machines, are condensed and collected by suitable machinery, much as in gas-works.

In this country it has been tried by several large companies. The yield of coke is greater than that of a bee-hive oven, but not more than that of a Coppée or Appolt oven. Considerable quantities of tar, &c. have been collected, but the quality of the coke is not so good, and the expense in labour and repairs is greater. As the quality of the coke is a matter of great importance in iron smelting, it is not surprising that this method of making it has been practically abandoned in this country; an additional reason is to be found in the fact that little or no real saving could, at any rate until recently, be effected, as the tar, ammonia, &c. realised very low prices, owing to the large quantities produced in gas-works. The prices of these bye-products have, however, within the last few years risen considerably, owing to increased demand.

On the Continent the Pernolet system has met with more favour, but there is no reason to believe that the coke there produced is better in quality than that obtained here. Ovens of all sorts of shapes have been tried. As the result of two years' experience by Messrs. Bell Bros. in the North of England, preference was given to the bee-hive shape ; but the process was finally abandoned, mainly, it is said, owing to the flues and floors being constantly burnt down by the excessive heat of the gases.

The manufacture of good hard dense coke is incompatible with the production in any *considerable quantity* of valuable condensable gaseous compounds. This follows from the established fact that the higher the temperature, and, within certain limits, the longer the coke is submitted to that temperature, the greater will be the yield, and the more dense the coke. At low temperatures volatile bodies, rich in carbon, are formed, and may be collected ; at higher temperatures these compounds are decomposed and deposit their carbon. Thus, the substances which distil off from the central cooler region of a retort will, on coming in contact with the outer layer, be decomposed, provided the temperature be sufficiently high. A temperature which is not sufficiently high to effect the decomposition of the richer hydrocarbons is below that necessary to produce a good dense and hard coke. In the manufacture of good *oven-coke*, so called to distinguish it from *gas-coke*, the gas produced cannot be highly luminous, for at the temperature required to produce such coke olefiant gas, the principal light-yielding constituent of gas used for illuminating purposes, would be decomposed.

From what has been said it will be gathered that, provided the proper conditions for obtaining a hard dense coke be attended to, there is no real objection to the collection of such tar and ammonia as may then be formed ; but these substances must always be of secondary importance, attention being mainly directed to the regulation of the process

so as to produce sound coke, the conduct of the process having to be varied somewhat for each kind of coal.

The carrying out of these conditions is the object of the *Carvès* system. The general arrangement is something like that of a Coppée oven, with the difference that the gases, although burnt on the exterior of the retorts, are first passed through a series of pipes cooled by water, forming a condenser, in which the condensable products are deposited; the gases are next passed through *scrubbers*, or vessels containing moistened coke, which remove the ammonia, the liquor when saturated being run off into reservoirs and subsequently treated for the preparation of ammoniacal compounds. The gases, after undergoing this treatment, are admitted through a nozzle in a fire-door to a fire-grate, on which a small amount of refuse fuel is kept burning to ignite the gases and air which has been allowed to mingle with them in suitable quantity. From the fire-grate the burnt and burning gases pass through the various flues surrounding the retorts. The gases, after they have done their work in the ovens, are further utilised for raising steam. It is stated that at Bességes all the machinery required in the manufacture of coke and its bye-products is now being driven by steam raised in this way, and there remains a large surplus, which is used in the blowing-engines for the Bessemer process, and for lifts, cranes, &c. The steam raised by this surplus heat is sufficient to furnish 400 horse-power. These ovens, we are assured, give a yield of 75 per cent., as against 55 to 65 per cent. obtained in an ordinary bee-hive oven; the coke is also freer from ash, since, the yield being greater, the ash is diffused through a greater amount of coke in the one case than in the other. The production of every ton of coke is said to be accompanied by the formation of about four shillings' worth of bye-products.

Good coke has been referred to as hard and *dense*. It must be understood that there is a practical limit to the

value of density ; it would not do, for instance, to compress it by artificial pressure.

The most economical and generally satisfactory coke-ovens yet produced are unquestionably those based on the principles embodied in the Coppée, Appolt and Carvés systems, where all spare heat is utilised for raising steam. There are yet many improvements to be made in applying these principles in practice.

GASEOUS FUEL.

The theoretical calorific values of carbonic oxide and of hydrogen have already been fully discussed ; it remains to show in what way these gases can best be turned to practical account.

It will be recollected that the intensity of combustion of hydrogen and of carbon, when burnt in air, is practically the same, whilst the heat of combustion of carbonic oxide in burning to carbonic acid is somewhat greater than that of carbon likewise oxidised to the maximum. So far, then, as the actual intensity obtained on combustion is concerned, it would appear that it matters little whether we employ solid or gaseous fuel ; in practice this is very far from being the case. When the fuel is gaseous it is much easier, for instance, to apply heat uniformly to a given surface, or locally, as may be required in some cases, and further, the regulation of the temperature is much more within control, and complete combustion can be ensured. This is well illustrated by the system of heating coke-ovens constructed on the principle of the Coppée and Appolt. Another advantage, which is of great importance, is that the gases can be generated in any convenient place, and led by pipes, or other suitable means, to the spot where their combustion is required to be effected. In this way valuable space can often be economised, and working room obtained where it is most required.

These are very far from being the only advantages which may be derived from the use of gaseous fuel. When great intensity of heat is required it is possible to produce it by heating the gases and the air required for their combustion ; and, if desired, the gas and air can be burnt under pressure, thus still further increasing the temperature obtainable ; the only limit being that fixed by dissociation. In practice these high temperatures could not be obtained by simply heating solid fuel and the air supplied for its combustion, though there is, up to a certain point, a very decided gain in heating the air.

Strictly speaking, all heating operations are conducted, to some extent at least, by means of gaseous fuel, for all fuels yield on distillation gases, with the exception, practically, of coke and anthracite ; and, apart from the products of distillation, it is essential to the manner in which combustion is effected that carbonic oxide should be formed, which subsequently burns on being brought into contact with air on the exterior of the heated fuel. Thus, in reverberatory furnaces, in which the fuel is burnt in a chamber separated from that in which the smelting or other operation is to be effected by a low wall, above which the two chambers communicate, the heating of the bed of the furnace is really accomplished mainly by gaseous fuel.

The products due to distillation of the upper and cooler fuel, and the carbonic oxide derived from the decomposition of the carbonic acid, first formed by the admission of air under the grate, are drawn by a chimney at the opposite end over the partition-wall or *firebridge*, and, becoming mixed with a suitable quantity of air, are deflected down on to the hearth. The relative amount of combustion which takes place on the hearth and above the fuel in the grate will depend on the quantity of air allowed to enter the opening through which the fuel is charged in. The principle of the modern gas-producer is essentially the same as that just described ; important modifications in construction are

even yet being made. The gas-producer first assumed importance on the introduction of the Siemens' 'regenerative' system in 1856.

The arrangement then adopted (Fig. 11) is still, with little exception, the only one in use. It consists of a chamber lined with fire-brick ; one side (B) slopes at an angle

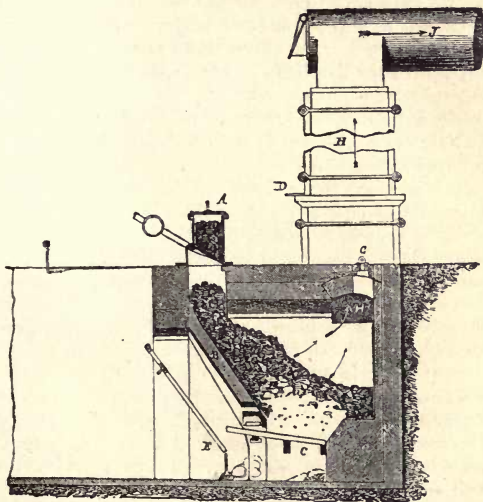


FIG. 11

of from 45° to 60° ; at the bottom is the grate (c). The fuel is introduced into the hopper (A), the cover is replaced, and the bottom having been lowered by means of the weighted lever attached to it, the fuel falls on to the incline, down which it gradually moves to supply the place of that which has been converted into gas in the region of the

grate. Air is admitted only through the grate. The operation is exactly the reverse of that which takes place in a beehive oven. In this producer, the air being admitted at the bottom, combustion proceeds from below upwards, and the volatile portion of the fuel is distilled off without coming in contact with air, whilst that which in the coke oven would become coke is converted into carbonic oxide gas in the producer. A small quantity of water is admitted by the pipe (E) to the ash pit, where it evaporates, and, becoming decomposed by the incandescent fuel, enriches the producer-gas by the formation of carbonic oxide and free hydrogen. By means of the plug-hole (G) the interior of the producer can be inspected, and, if necessary, the fuel stoked with an iron rod.

It is important that a slight outward pressure should be maintained in the flue, to prevent the gas being partially consumed by an indraught of air through crevices in the brickwork. If the furnace could be placed about 10 feet higher than the producer the required pressure would at once be obtained ; in practice this arrangement is rarely convenient. The following plan has been adopted. The gases, which on leaving the producer have a temperature of about 400° C., pass to the uptake (H), rising 10 to 20 feet vertically ; thence they traverse the horizontal iron *cooling-tube* (J), which may with advantage be several hundred feet long ; at the end of the cooling-tube the gases, having then a temperature of below 100° C., descend a tube called a *down-comer* into a main, whence they are drawn off as required. The effect of cooling the gases is to increase the relative weight of a given volume 50 to 60 per cent. Thus a sort of siphon action is set up, the heavier gas in the down-comer representing the longer leg of a siphon ; at the same time, by this arrangement an outward pressure is maintained in the flues of about $\frac{1}{10}$ th to $\frac{2}{10}$ ths of an inch of water more than that of the external atmosphere. By cooling the gases aqueous vapour is condensed and deposited.

Although the desired results are obtained in this way, yet it will be observed that it is only arrived at by the dissipation of a great deal of heat. Were this, however, not so, a chimney or an exhaust-engine would be required to draw the gases from the producer, so this heat must be looked upon as doing important work. The heat withdrawn from the cooling-tube, instead of being carried off by the surrounding air, might be imparted to water for raising steam under boilers ; such arrangements, very simple in principle, are generally extremely difficult to carry out satisfactorily in practice, each particular case requiring special treatment, involving numerous and expensive experiments, which, after all, may not lead to success.

It has been stated that steam may with advantage be admitted with the air supplied for combustion. The advantage which steam possesses over air is due to the large quantity of inert nitrogen in the latter. Nevertheless, it must not be forgotten that in the decomposition of steam by carbon heat is *absorbed* ; if, therefore, much steam were admitted, the temperature would eventually be so much lowered that combustion would only proceed with the greatest difficulty, if at all.

Up to a certain amount steam may with advantage be admitted ; beyond that it does harm. It is very important that this should be properly understood, since a great deal of misconception exists as to the use of steam. It should only be introduced when there is an excess of heat in the furnace beyond that required to carry on with sufficient rapidity the conversion of the solid fuel into gases.

That heat is absorbed during the reduction of aqueous vapour by carbon is easily shown. One part by weight of hydrogen, in combining with eight parts of oxygen to form water, evolves about 34,000 units of heat, and exactly the same number of units will be absorbed again on the water being broken up into its constituents. The eight parts of oxygen liberated will combine with carbon, evolving 14,838 units,

since carbonic oxide is the final product so far as the gas-producer is concerned ; some of the hydrogen will remain in the free state, the remainder unites with carbon to form, mainly, carburetted hydrogen gas (CH_4), evolving about 16,000 heat units for each unit of hydrogen. Supposing, for the sake of argument, that the whole of the hydrogen of the decomposed steam combined with carbon to form carburetted hydrogen gas, and the oxygen to form carbonic oxide, then the total heat units absorbed and evolved would stand as follows :—

Absorbed by decomposition of 8 pts. H_2O	=	34,462	heat units.
Evolved by combination of 1 pt. H + 3 pts. C	=	16,000	„ „
„ „ „ 8 pts. O + 6 pts. C	=	14,838	„ „
Total evolved	=	30,838	„ „
Balance in favour of absorption	=	3,624	„ „

The following analysis of gas made in the producers at the glass-works at St. Gobain, with a closed grate and steam-injector, will give an idea of the amount of hydrogen which may remain in the free state, as compared with that carburetted :—

	Volume	Weight
Carbonic oxide . . .	24.2	25.8
Hydrogen . . .	8.2	0.6
Carburetted hydrogen . . .	2.2	1.4
Carbonic acid . . .	4.2	7.0
Nitrogen . . .	61.2	65.2
	<hr/> 100.0	<hr/> 100.0

One-fourth by weight of the carburetted hydrogen consists of hydrogen, which in this case will therefore amount to 0.35, or about half that in the free state. The whole of the hydrogen present appears an insignificant quantity, being only about 1 per cent.; but it must be recollected that every one part of hydrogen due to the introduction of steam means that eight parts of oxygen have been made available for combustion without nitrogen accompanying them, *i.e.* there will be $8 \times 3.33 = 26.64$ parts less nitrogen than if air con-

taining an equivalent quantity of oxygen had been used. It is not intended to imply that the whole of the hydrogen in this particular case was introduced by the steam: some of it was probably derived from the coal.

In the form of producers just considered, a good deal of the hydrogen originally present as a constituent of the coal is removed in combination with carbon, as *tar* and hydrocarbons, which condense in the flues, and are a source of inconvenience. The most recent forms of producers proposed by Dr. Siemens have for one of their principal objects to consume the tar, &c. in the producer.

Fig. 12 illustrates the most recently patented (May 1882) producer of this latter kind. It may be worked with or without a steam-jet, as may be required. The air is admitted through openings, as shown at N, Fig. 12, into a chamber (L), constructed in cast iron, and having pockets (M); thence it is drawn off through channels (P), by the action of the steam-jet (Q), supplied through the pipe (R). S is a grating, and T a pipe through which water is supplied to an annular perforated pipe, shown in section, by means of which a spray is projected on to the clinker and ashes below. The fuel is charged in through K, and the gas passes up through the channels (U) into the main (I). The important point about this form of producer is that the volatile products of distillation, instead of passing out through the cool fuel at the top—as in the ordinary arrangement, whereby tar, &c. are deposited in the flues—are obliged to *descend through highly heated fuel*, and are thus completely decomposed before reaching the flues. The *spare heat* is utilised to decompose steam and heat the air. The gases as they leave the producer pass round the pockets (M), strongly heating the air contained in them, which then, being specifically lighter, rises, and is drawn off through the channels (P), by the action of the steam-jet, whilst cooler air sinks into the pockets, and so on. The pockets may be replaced by coils of pipe.

The Wilson gas-producer, which is being used in many

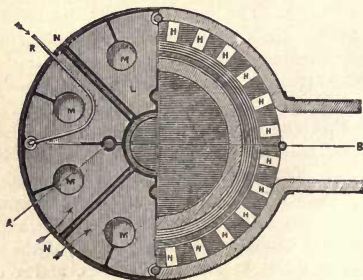
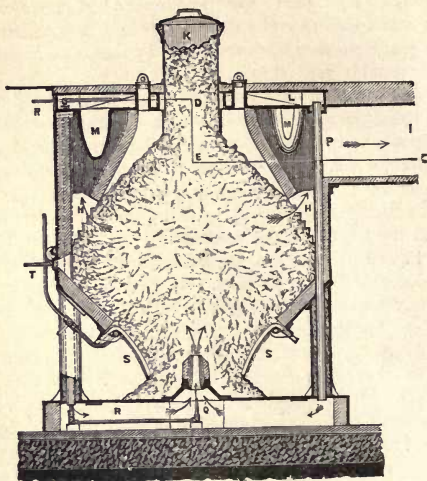


FIG. 12.

works, and the Dowson gas-producer, which has been successfully applied to the driving of machinery by means of a gas-engine, are of the last type.

The idea of heating, by what is now known as the regenerative system, the air supplied for the combustion of *solid* fuel occurred to Robert Stirling in 1816. The method by which he proposed to carry it out, as set forth in his patent, is very crude, and would have been in that form impracticable. Still it embodies the notion which forty years afterwards was again conceived by Messrs. Frederick and C. W. Siemens, in whose hands it has brought about results of incalculable value. They, like Stirling, began with the idea of heating air for the combustion of solid fuel, the general arrangement in both cases being very similar. There was to be a single fire-place, supplied with fuel from a hopper above it, two chambers to contain the substance to be heated, and at the end of each chamber a regenerator. The regenerator in the Siemens arrangement consisted of chequer-work of brick, *i.e.* a chamber packed with bricks so placed as to leave spaces, through which the spent gases had to pass on their way to the chimney. Stirling proposed to use thin partitions of metal or glass. In either case the object was to cause the gases to give up their heat to the cooler fire-brick or other material.

When the regenerator through which the products of combustion had been made to pass had become sufficiently heated up, the direction of the current was reversed, and the air as it entered then became heated by the hot masonry which it had to traverse, whilst the other regenerator in its turn had its temperature raised again by the products of combustion. Reversal of the direction of the currents was to be effected at such intervals as would prevent the regenerator through which the air was passing becoming cooled to too low a temperature. The conditions of this arrangement necessitated that the operation being carried on in each furnace should be completed in the time between the reversals—a

state of things at the best extremely inconvenient, and for most purposes altogether impracticable.

It was at this point that the brothers Siemens conceived the idea of employing gaseous instead of solid fuel, and heating it *as well as* the air by means of regenerators. It would be difficult to over estimate the importance of the results which have followed the introduction of this system. By its means the highest attainable temperatures can with ease be produced, the only limit being that at which dissociation takes place.

Furnaces on this system have been applied with very great success to the manufacture of soft steel, and for reheating iron and steel ; also to zinc smelting, and in gas, glass, pottery, and porcelain works. It is not too much to say that there is no purpose for which a high temperature is required for which it might not with great advantage be employed, in respect both to general convenience and economy.

Figs. 13 and 14 represent the furnace, the one being a longitudinal and the other a cross section ; the gas coming from the gas-producer passes in through the flue (H) and the reversing valve (F), by means of which it is directed into the bottom part of the regenerator chamber (C), on the left. The gas flowing up through the mass of brickwork the chamber contains, and which is placed so as to form a large aggregate surface, with intricate zigzag passages, will become heated, provided any heat has been accumulated therein. In the first place there will be no heat, and the gas will pass unheated through this chamber and thence to the combustion-chamber of the furnace (A). At the same time, a current of air is admitted through the air-reversing valve (E), into the air regenerator chamber (D), which is larger than the gas-chamber. The air passing up through the chequer-work will reach the same point as the gas does at the entrance into the combustion-chamber of the furnace. Now, since both the air and gas are cold, and as they meet for the

F

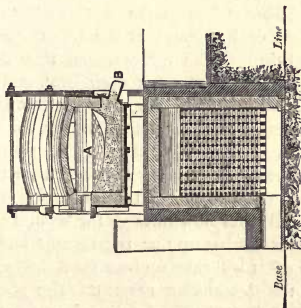


FIG. 17

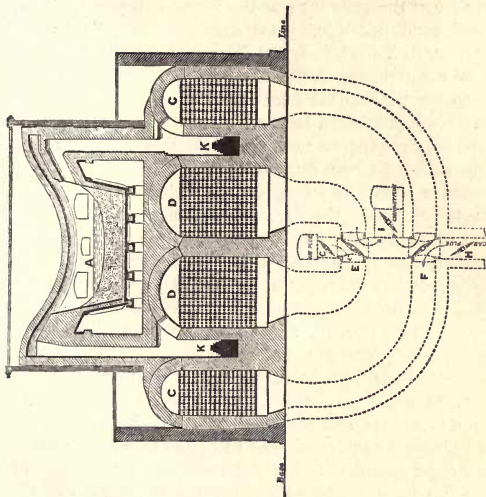


FIG. 14

first time at the entrance into the furnace, they will, if there ignited, produce a heat not certainly superior to what would be produced if solid fuel had been burned there instead ; on the contrary, gas of the description we are dealing with is a poorer fuel than solid fuel, and the heat produced in the furnace will, therefore, be very moderate indeed. But the flame, after passing over the bed of the furnace, does not go to the chimney direct, but has to pass through the two regenerative chambers (D, C), on the right, similar to those already described ; the larger proportion of the heated products of combustion will pass through the air regenerator chamber, simply because it is the largest channel, and another portion will pass through the gas regenerator. The products of combustion pass from these chambers through the reversing valves (E, F), and are by them directed into the passage (I) leading to the chimney.

The operation, therefore, is simply this : that the air and combustible gas pass up into the furnace through the one pair of chambers, and pass away, after combustion, towards the chimney through the other pair ; but in passing through the second pair the heat of the products of combustion is given up to the brickwork. The upper portions of this brickwork take up the first, and, therefore, the highest degree of heat, and, as the burnt gases are passed downward through the regenerators, they are, by degrees, very completely deprived of their heat, and reach the bottom of the chambers and the chimney comparatively cold. After this action has been going on, say, for an hour, the reversing valves are turned over. They are simple flaps, acting like a four-way cock, and, by throwing over the levers which work them, the direction of the currents is reversed. The gas and air will enter now through the second pair of chambers, and the air passing up one regenerator and the gas passing up the other will take up heat from the bricks previously heated by the descending current. The gases so heated, say, to 1,000° F., will enter into combustion, and if the heat pro-

duced at the former operation was $1,000^{\circ}$, it ought this time to be $2,000^{\circ}$, because the initial point of temperature is $1,000$ higher. The products of combustion will also escape at $2,000^{\circ}$, and passing through the chequer-work of the first pair of regenerators, its uppermost ranges will be heated to very nearly $2,000^{\circ}$. The temperature will diminish by degrees in descending till the gaseous currents have again reached the bottom nearly cold. Again reversing the process, after another hour or half-hour, as the case may be, the gas will take up heat to the extent of nearly $2,000^{\circ}$, and since another $1,000^{\circ}$ is again produced in combustion the temperature of the furnace will this time attain $3,000^{\circ}$; and in this way it might be argued that, unless work is done in the furnace, the heat developed in combustion will, step by step, increase the temperature of the furnace, $1,000^{\circ}$, or something less, each time a reversal of the valves takes place, till we arrive at the practical limit imposed by the melting-point of the most refractory substance we can find (pure silica, in the form of Dinas brick), of which the melting chamber is usually formed. This high temperature is obtained by a gradual process of accumulation, and without any such current as would be likely to destroy, by oxidation, the metal in the bath, or cut away the sides and roof of the melting-chamber.

There is, however, a theoretical as well as a practical limit to the degree of heat obtainable in combustion, which was first pointed out by M. H. St. Claire Deville, namely, the point of dissociation at which carbonic acid would be converted back into its constituents, carbon and oxygen. If carbonic oxide, or any other combustible gas, and air enter the furnace at a temperature very nearly equal to the point of dissociation, it is evident that association or combustion cannot take place, and thus nature fortunately steps in to restrict the increase of heat by accumulation within comparatively safe limits. In a furnace fully heated up to the melting-point of iron, this action of dissociation can be very

clearly observed. At first, when the gas and air are comparatively cold, combustion takes place sluggishly, the gases will flow through the furnace and produce only a dark-red flame ; the next time the valves are reversed a whitish flame is produced ; the next time, a short white flame ; and after having reached a full white heat, exceeding the welding-point of iron, the flame will again become a long one, but this time not red and of little apparent power, but bluish-white and flowing in clouds. This indicates the near attainment of the point of dissociation ; combustion can no longer take place, except in the measure of the heat being dispersed to surrounding objects or to the metal in the furnace, and that is about the degree of heat required for the process of making steel on the open hearth.

The regenerators just described are only well suited for use when very high temperatures are required. For comparatively low temperatures the following arrangement, which has been patented several times during this century, is better suited, though up to quite recently there have been difficulties in the way of carrying it into effect. A parallel series of long narrow passages are arranged so that the hot products of combustion may be passed through alternate passages, the intervals thus left to serve for the introduction in the *opposite direction* of the air to support combustion. In this way the heat of the outgoing gases would be communicated to the air and carried back into the furnace. It was at first proposed to make the partitions of sheet iron ; this, however, did not prove satisfactory, owing to the leakage and the inability of the partitions to stand the wear and tear consequent on the high temperature. Fire-brick partitions were substituted for iron, but with no better success, as the air and gases mingled through the cracks and interstices. The objection to the use of fire-brick is said now to have been satisfactorily removed by the simple device of glazing the surface of the partitions. If this be the case, then regenerators on this system have a wide field

open to them. They cannot be employed when a very high temperature is required, as the heat is not accumulated, and the temperature of the air can never be higher than half that of the outgoing gases.

Although the use of iron partitions was not successful, yet in a modified form this system has since been largely employed in the manufacture of iron. The air to be heated is passed through a series of cast-iron pipes, arranged in a chamber, through which the waste gases, as they are termed, from the blast-furnace are made to circulate. There is, however, an important distinction between these gases and those obtained from reverberatory furnaces. In the latter case they have practically undergone complete combustion, whereas in the former, owing to the combustion being effected from the bottom of the furnace, and the products having to pass through incandescent fuel before escaping at the top, and for other reasons connected with dissociation, the gases contain a very considerable quantity of carbonic oxide, which is capable of producing, on combustion in the *stove* for heating the pipes, a temperature very much higher than that which could be obtained were only the sensible heat being carried off by the gases available.

The gases collected from the top of blast-furnaces contain on the average about 25 per cent. by volume of carbonic oxide, and hydrogen varying in quantity, according to circumstances, up to 7 or 8 per cent. From this it will be seen that, in point of heat-evolving power on complete combustion, blast-furnace gases are quite equal to producer gas. Until within the last few years these gases were allowed to burn uselessly at the top of the blast-furnaces; now they are generally—even yet not universally—collected, and employed for heating the blast and raising steam; in many works no other fuel being required for such purposes.

The use of iron pipes has always been attended by a

serious drawback, viz. that if by carelessness the temperature was allowed to get up above $1,000^{\circ}$ F., the pipes would be ruined, so that practically about 900° F. is the maximum temperature which it is safe to work with ; more commonly 600° to 800° has been employed, until quite recently. Another objection is that leakage is liable to occur, owing to the unequal expansion and contraction of the different parts of the stove-pipes.

Soon after the introduction of the regenerative system by Dr. Siemens, Mr. Cowper proposed to construct a stove on that principle, as a substitute for the iron-pipe stoves. Since their introduction these stoves have been modified in several ways, chiefly by increasing their height and arranging the passages so as to allow the gases a free passage, and at the same time to withdraw the main part of their heat.

Figs. 15 and 16 show the stove in its present form. The gases are drawn off from the top of the blast-furnace by a tube called a down-comer, and passing into the gas flue *v*, enter the stove by the valve *F* ; by means of the valve *G* a suitable quantity of air is admitted, which, mingling with the gases, effects their complete combustion in the flame-flue *o*, from the top of which the highly heated products pass into the regenerator *P*, consisting of a long column of chequer-work ; from the bottom of the regenerator the gases are drawn off by the chimney, through the flue *u*, the draught being regulated by the valve *D*.

The whole of the stove, with the exception of the valves, is enclosed by several courses of fire-brick, encased in iron. As the gases pass through the regenerator they give up the greater part of the heat they contain to the top layer, there being a gradual shading off, as successive courses of bricks are passed through, until the bottom is reached, when they will be found to have a temperature of only about 300° to 400° F. It should be borne in mind that some heat must be left in, to create a sufficient draft in the chimney. The temperature of the gases issuing from an iron-pipe stove

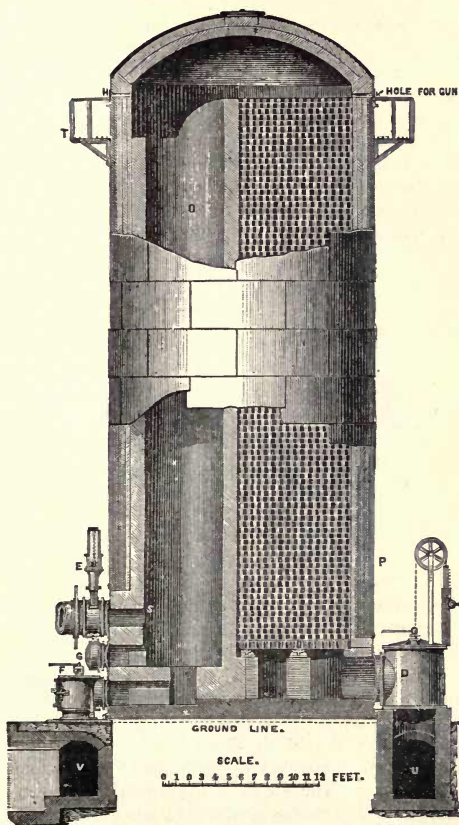


FIG. 15

is, however, between $1,200^{\circ}$ and $1,300^{\circ}$ F., so that a considerable waste of heat takes place. So soon as the top layer of the regenerator has become heated up to the temperature of the gases passing through it, it must of necessity cease to absorb heat, and the gases traverse it without undergoing any alteration until they arrive at a part of the brickwork at a lower temperature than themselves, which they then gradually raise to their own temperature. If the gases were passed through the stove for some time, the whole of it would eventually become of the same temperature as the

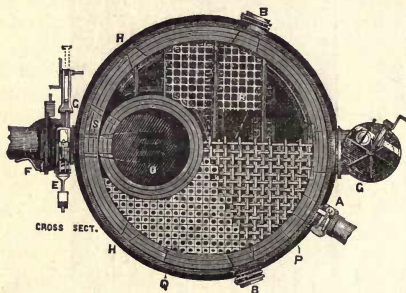


FIG. 16.

gases, which would then issue at the same temperature as that at which they entered. In practice, to avoid loss of heat, it is desirable to stop the heating up of the stove whilst the maximum temperature is still some way from the bottom ; the gas and air are then admitted to another stove, and the chimney-valve of the heated stove having been closed the blast of cold air which it is desired to heat is passed through it, entering by the valve at the bottom of the regenerator. The air, as it traverses the regenerator, becomes heated to the temperature of the stove, and, passing down the red-hot flame-flue, is conducted to the blast-furnace

through the valve E. A peep-hole through which to observe the temperature of the blast is provided at S. By means of this stove the blast can be heated to $1,500^{\circ}$ F., the reduction in temperature after blowing cold air through it for about three hours amounting only to about 100° , and 150° in four hours, as the air on entering soon becomes heated to the temperature of the stove, and then passes on without withdrawing any heat from the remainder of the regenerator above, so that the temperature continues constant as long as a certain area of the regenerator remains at its initial temperature. Mr. Cowper states that the saving of fuel due to the use of these stoves amounts to 20 per cent., and in some cases is as much as $7\frac{3}{4}$ cwt. of coke per ton of iron made. Dust is, as far as possible, by a special arrangement of the pipe supplying the gases, prevented from entering the regenerator ; that which finds its way in is removed every few months by turning on the cold blast, and shutting all the other valves ; a special door is then opened several times : the sudden rushing out of the compressed air dislodges and carries out the dust. In some works the same result is arrived at by firing into the stove, through openings provided for the purpose, a gun loaded with blasting powder ; the dust collects at the bottom, and is removed through doors. The friction is very much less in these stoves than when pipes are used ; the power necessary to produce a blast of the required pressure is therefore also less. The height and diameter of these stoves varies, the former up to 55 feet, the latter to 25 feet.

Fig. 17 shows a modification of the Cowper stove, known as the Whitwell stove. The arrangement is intended to facilitate cleaning. It is only fair to say that the Cowper stove, as now made, is said by many competent to give an opinion to present no special difficulty in this respect. Both stoves are capable of heating the blast to as high a temperature as is at present employed in blast-furnace practice ; they are both largely used, and not uncommonly are seen side by

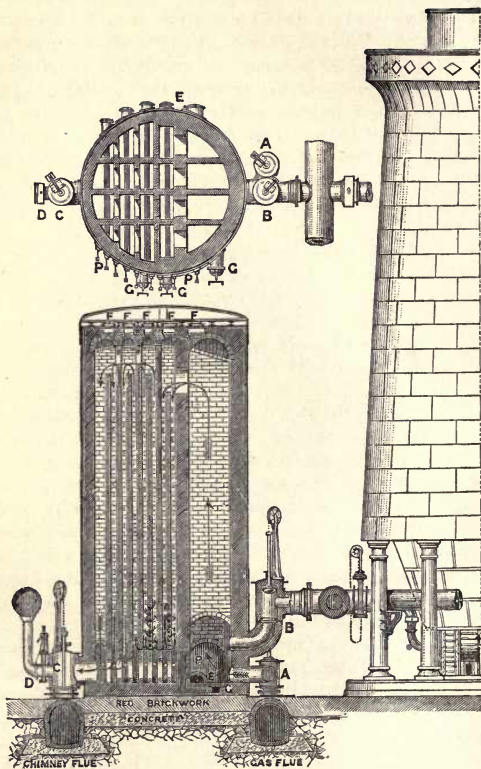


FIG. 17.

side in the same works. The figure shows the attachment to the blast main of the furnace. A is the gas valve, G the air inlets for the combustion of the gas. The air is warmed by being passed through the basement of the stove, which it thus protects from damage by excessive heating. C is the chimney-valve. It will be observed that the chequer-work of the Cowper stove is replaced by fire-brick partitions. At the bottom of the first and third of the smaller partitions are openings, through which a further quantity of air is admitted to complete the combustion of the gases.

The cold blast enters through D, and passes out heated at B; P P are the eye pieces through which to observe the temperature. In order to clean the stove, the gas is shut off, and the chimney-valve (C) slightly opened in order to carry the heat downwards, the first top cleaning door (F) is taken off, and the movable plug in the crown of the arch lifted. Scrapers with $\frac{3}{4}$ -inch tubular handles are introduced, and the walls scraped down, the dust falling to the bottom; the door is then replaced, and the next compartment treated in the same way, whilst the walls are still red-hot, and so on in rotation. The dust which has been detached from the walls is removed at the bottom through the doors (E). These stoves, in America, have been made as much as 70 feet high and 21 feet in diameter, containing 30,000 square feet heating surface.

REFRACTORY MATERIALS.

THE term REFRACTORY is necessarily relative, the degree to which a substance is refractory depending on the circumstances under which it is employed, and notably on the nature of the substances with which it is brought in contact. For example, silica (SiO_2) heated alone, even to very high temperatures, is extremely refractory, *i.e.* it does not sensibly soften, but if it be brought into contact with some base, such as lime (CaO), oxide of lead, or oxide of copper, at an ordinary red heat, then union takes place with the formation of a silicate, which in every case is much more fusible than silica, though differing in degree according to the base. It is obvious from the foregoing that in the selection of materials for the construction of furnaces the chemical affinities which will be brought into action between the body of the furnace and the substances to be treated in it must be carefully considered. This, however, is not the only point to be borne in mind in choosing a suitable material. Many substances are capable of resisting a very high temperature if *gradually* raised to it; but have not the power of resisting sudden changes of temperature, such as occur, for instance, when large crucibles have to be removed from a furnace in order to discharge their contents, or when a crucible is heated suddenly by placing it in a hot furnace. For these purposes such materials would be unsuitable, and it would be necessary to select some substance or combination of substances capable of resisting sudden alterations of temperature.

The most important refractory material is that resulting from the so-called burning of clay. We will, therefore, first consider the composition of various kinds of clays, and the properties which give it so great a value in the arts.

The basis of clay is silicate of alumina, *i.e.* silica in chemical combination with alumina. All clays have been derived from the natural decomposition of felspars, which occur in nearly all igneous rocks and in some stratified crystalline schists. The character of each clay will be mainly determined by the nature of the rock from which it originated. The felspars consist essentially of anhydrous silicates of alumina, with variable proportions of silicates of potash, soda, lime, and magnesia, and a little oxide of iron. The relative proportions of these alkaline bases is found to vary with the state of occurrence of the felspar ; thus, orthoclase and the other more silicious felspars containing potash are common in granite and all plutonic rocks, whilst the volcanic rocks are characterised by the presence of the less silicious felspars, containing soda and lime. By 'weathering,' *i.e.* exposure to the combined action of carbonic acid and water, the soda, potash, lime, and magnesia are dissolved away more or less completely, and at the same time the residual silicate of alumina becomes chemically combined with water, and is said to be hydrated.

In its purest state clay would consist of silica, alumina (Al_2O_3), and water, combined in proportions probably in most cases represented by the following formula : $\text{Al}_2\text{O}_3 \cdot 2 \text{SiO}_2 + 2 \text{H}_2\text{O}$, which corresponds to a percentage composition of 46.33 SiO_2 , 39.77 Al_2O_3 , and 13.90 H_2O . This state of purity is closely approached in china clay, used in the production of the best porcelain. The white scaly, crystalline powder of which china clay consists, seen through the microscope, is found to be made up of flexible but inelastic hexagonal plates. These plates, with the aid of the microscope, can be discovered in greater or less quantity in every clay.

The degree of purity of a clay will depend not merely on the extent to which the felspar has been decomposed, but also on the more or less completeness with which it has been separated from the containing rock by the action of running water. Thus, clay derived from orthoclase, which

occurs in granite, will be found more or less admixed with quartz and mica. All clays contain more or less admixed silica.

It will have been remarked that during the decomposition of felspar the silicate of alumina has passed from the anhydrous to the hydrated condition. It is to this, at first sight, unimportant change that clay owes its great value, for it is in consequence of the presence of this water of hydration that clay possesses its characteristic property of *plasticity*, or the capability of being moulded into any required form when moistened. If dry clay be moistened it can readily be made to take any required shape, and if then it be dried by the warmth of the sun, or by some similar moderate heat, it will preserve its shape, so long as it is not remoistened. If, on the other hand, the clay be subjected finally to a fairly strong heat, it will no longer possess the property of plasticity on being remoistened : it will be rigid, and practically a totally different substance to that from which it was derived. This change, so important in its consequences, is due simply to the fact that at the higher temperature the water in chemical combination has been removed, whereas at the lower temperature the mechanically admixed water alone was withdrawn. The chemically combined water once driven off never again resumes its union ; to this is due the stability of the innumerable structures made from clay.

Since, then, clay has been derived from the gradual decomposition of certain portions of the earth's crust, and has in most cases, by the action of running water, been accumulated as a bed or stratum over large areas, it is easy to understand how it is that considerable variation in its composition and state of division may occur, even within a comparatively limited space.

Clays are divisible, broadly speaking, into two classes, viz. ordinary clays, such as are used for building houses and the cooler portions of furnaces, and fire-clays, which, as the name denotes, are capable of withstanding a strong heat

without rapidly deteriorating. The difference between the two kinds is due to the amount and nature of the impurities, *i.e.* foreign substances, which they respectively contain. The more potash or soda a clay contains the more easily fusible will it be. Oxide of iron, lime, and magnesia act in a similar manner, though to a much less extent.

The fire-clays, with which we are here more especially concerned, are obtained principally from the coal measures, in which they occur in great abundance. The neighbourhood of Stourbridge has long been celebrated for the quality of its fire-clay, which is largely used for the manufacture of bricks of various shapes, and for crucibles.

During the dehydration of clay by heat contraction takes place, the amount of contraction depending on the degree of heat applied. In bricks this shrinkage is not of so much consequence ; but in crucibles, owing to the variation in thickness and shape, the material would be liable to crack. In order to counteract this tendency a proportion of some substance which either expands on heating, or at any rate does not contract, is added. For this purpose burnt fire-clay is commonly mixed with the raw clay, and sometimes coke dust, graphite, or silica, which last expands on heating. When burnt clay is used, a common proportion is one-third burnt clay to two-thirds raw clay. The amount of burnt clay necessary is, however, very variable, depending on the 'fatness' of the raw clay. It is important that the burnt clay should not be too finely powdered ; the size of the particles will depend on the use to which the material is to be put. The presence of particles of iron pyrites, of any appreciable size, in fire-clay used for crucibles should be carefully guarded against, as the oxide of iron, resulting from the oxidation of the pyrites under the influence of heat, combines with the clay, forming a fusible substance, and causing a hole to form at that point.

The power of any clay to resist a high temperature may be judged of by working up some of it with a suitable quantity

of water, shaping it into a rectangular piece with *sharp* edges, and then, after drying it at a gentle heat, subjecting it to the temperature which it will be required to stand, care being taken to protect the clay from the action of the ashes, if solid fuel be employed for obtaining the requisite heat. If the edges have remained quite sharp, the clay may be considered highly refractory at the temperature of the experiment.

The manufacture of large crucibles is conducted in one of two ways. In the old method, and that still in general use, a mixture of various clays which experience has shown to be best for the purpose is ground under edge-stones, and, having been sifted, it is mixed with water and the proper proportion of burnt clay, for which the old pots are utilised. This mixture is then kneaded, by men treading barefooted, until it has acquired the requisite consistency.

The crucibles are fashioned by hand, with the assistance of a block of the form of the inside.

Crucibles are now also made successfully by machinery. A sufficient quantity of the substance to be made into a crucible is dropped into a mould, which is then made to rotate; a fixed wooden tool, which descends nearly to the bottom of the mould, presses the lump and causes it to take the required shape. The moulds are made of plaster of Paris, supported when necessary by a wooden casing on the outside, in which case both the wood and the plaster of Paris are made in segments. The mould containing the freshly made crucible is placed in a warm place; in a short time, owing to the plaster of Paris absorbing the moisture from the surface of the crucible, the latter readily detaches itself.

Clays vary considerably in their power to resist the corroding action of metallic oxides, such as oxide of lead or oxide of copper. This is often a matter of considerable importance, and is best ascertained by direct experiment on a sample. No clay can resist the action of these oxides in their uncombined state for very long; some sulphides, *e.g.*

galena (sulphide of lead), have the singular property of readily permeating clay crucibles without corrosion.

The finer and more regular the grain of a crucible the more readily will it, as a rule, resist corrosion and permeation ; but unfortunately these same conditions act unfavourably in another way, by increasing the liability to crack.

Small crucibles are fired in kilns, and are allowed to cool down, as they can be heated up again without danger of their cracking. This cannot be done with large crucibles, such as are used for melting steel or large quantities of brass. They are first carefully air-dried, and then kept in a warm dry place until required for use, when they are gradually heated up to redness, *mouth downwards*, and are, after being inverted, ready for immediate use. It has been found that if a crucible receive its first heating mouth upwards it almost invariably cracks.

In cases where an extremely high temperature has to be resisted, *e.g.* in a Siemens steel-melting furnace or a Bessemer converter, fire-bricks made of clay cannot be employed, and some material still more refractory under the conditions has to be substituted. For this purpose silicious bricks are employed ; bricks of this kind were not known before about the year 1822, when the method of making them was discovered. The rock of which they are composed occurs in considerable quantity in the Vale of Neath ; it consists of about 97 per cent. of silica, the remainder being made up of alumina, oxide of iron, lime, potash, soda, and combined water. It is locally known as 'clay.' The rock having been sufficiently disintegrated, is mixed with about 1 per cent. to 3 per cent. of lime and some water ; it is then pressed into moulds, dried, and fired hard for about a week. On one of these bricks being fractured, it will be seen to consist of angular fragments of quartz, embedded in a light yellowish-brown matrix. In setting these bricks a cement of the same material must be used. Bricks of this description are now also made by grinding up fine any pure silicious

rock or stone, and mixing about 1 per cent. of milk of lime with it.

The addition of extraneous material to the silicious rock is in no way analogous in purpose to the case we have recently had under consideration, *viz.* the mixing of burnt clay, silica, or coke dust with raw fire-clay. In the present case we are dealing with a substance devoid of plasticity, and which, being in irregular pieces of variable shape and size, can only be made into a solid mass of a required form by the assistance of some cementing material. The small quantity of lime used answers this purpose very perfectly, as at a high temperature it unites with silica, forming a refractory silicate, which cements the mass together. If much lime were present the material would be insufficiently refractory at high temperatures. The presence of mica and oxide of iron in more than traces is also objectionable. Complex silicates, *i.e.* containing more than one base in combination with the same silica, are more easily fusible than single silicates, the silicates of potash and soda excepted ; it follows that the extent to which a brick is refractory will depend more on the number and relative proportion of the foreign substances present than on the actual quantity of any one of them. A clay containing a certain quantity of magnesia, and no lime, would be more refractory than if part of the magnesia were replaced by an equivalent quantity of lime.

Another silicious material, known as 'ganister,' is largely used for lining Bessemer converters. The finest quality of this material is obtained near Sheffield, where it forms the under-clay of a thin coal-seam ; it also occurs in Lancashire, in the Newcastle coal-fields, in those of Yorkshire, as well as in Shropshire. This substance appears to have been deposited in shallow seas, just after the mill-stone grit, and before the soil upon which the coal-measure plants grew. Geologically, it is probably the equivalent of the silicious rock of the Vale of Neath. It differs considerably from all

other sandstones, except that from the Vale of Neath; the particles of which it is composed are very fine, it has a waxy fracture, and is very hard and tough. Unlike other sandstones, it has the power of binding together, when ground fine and mixed with water. A fairly good substitute for this material can be made from powdered pure silicious stone, mixed with a certain amount of aluminous clay.

Natural silicious stones are occasionally used for the hearths of blast-furnaces and such purposes. They should be carefully dried before being strongly heated. The same remark applies equally to all bricks and crucibles, however produced.

In some cases in which sand, or a mixture of clay and sand, would not resist sufficiently well the corrosive action of certain slags, a greater or less amount of coke dust or débris of charcoal is added. This mixture is termed 'brasque.' Crucibles for melting steel and brass are now commonly made of plumbago or graphite, mixed with a certain amount of fire-clay to give strength. Talcose slate and soap-stones are used in some localities in the construction of furnaces. Serpentine resists corrosion well, but cannot with advantage be exposed to very high temperatures. Bauxite, a mineral which occurs in considerable quantity in several localities in France, in pockets in calcareous rocks belonging to the cretaceous series, and which consists essentially of pisolitic oxide of iron in a matrix of alumina, the latter largely predominating, has a limited use for lining furnaces. It is also used as a source of the metal aluminium. Its composition is very variable, but on an average it would contain about 3 to 5 per cent. silica, 23 to 25 per cent. peroxide of iron, 50 to 60 per cent. alumina, and 10 to 15 per cent. water. It also mostly contains 2 or 3 per cent. titanitic acid. This material is highly refractory, owing to the aluminate of iron, which forms at a high temperature, being very infusible.

In one important metallurgical operation oxide of iron is used as a furnace lining, viz. in the puddling process.

Here, owing to the nature of the process, either silica or clay would be rapidly corroded and cause great waste of iron, whereas oxide of iron not only protects the masonry of the furnace, but takes an essential part in the process. Iron itself in some processes is also employed, being kept sufficiently cool for the purpose by the circulation of cold water or air at the back of it.

We have seen how, in clay, alumina in combination with silica forms a very refractory substance. Alumina itself is even still more refractory, but owing to the difficulty of obtaining it in considerable quantity it is not available for use on a large scale. The same remark has hitherto applied to magnesia (MgO), which is also highly refractory.

Lime, which is infusible at any temperature we can command, is, on the other hand, extremely plentiful. In order to produce it, it is only necessary to drive off the carbonic acid from chalk or limestone by the application of heat. The use of lime, however, as a refractory material for furnace purposes is rendered impossible, owing to the property it possesses of combining readily with water, with the result that it swells up, and falls to pieces ; so that, unless the lime be kept continuously at a high temperature, the structure composed of it crumbles to pieces by the mere action of the moisture in the air. For small operations, such as the fusion of platinum by the flame of an oxy-hydrogen blow-pipe, blocks of lime have found an application.

The want of some substance of the nature of lime which could be employed at high temperatures has long been felt. It is only recently that this want has been successfully supplied on a large scale. It has been found that magnesian limestone, which consist mainly of carbonates of lime and magnesia, unlike ordinary limestone or chalk, after it has been very strongly calcined can be reheated and cooled as often as required without crumbling. In consequence of this important discovery, iron ores, which

previously were useless for the production of soft steel, are now being largely employed. This portion of the subject will be discussed when speaking of the manufacture of steel.

The most successful method of employing the magnesian limestone is, first to fire it strongly, then to grind it up fine and mix it with pitch. The bricks made from this mixture are then strongly fired. There is, however, considerable difficulty attending the construction of a lining with these bricks, because they have to be set with similar material, which it is almost impossible to fire satisfactorily, and which is very liable in any case to work out of the joints, owing to shrinkage. This difficulty is increased by the shrinkage of the bricks themselves during firing, which gives them a more or less curved form. These objections have been successfully overcome by making the material, which has been strongly calcined, into a 'slurry,' or liquid mass, and running it into a mould of the required shape and size, then by a gradually applied heat consolidating it by removing the pitch ; or the mixture of magnesian limestone and pitch may be in such proportion that on the application of heat it will fuse, the pitch finally being decomposed when the heat is increased. In this case the mixture may be shovelled into the mould in a solid condition.

All sorts of magnesian limestone (*dolomite*) are used at different works ; but for material which is to be highly shrunk, and when it can be obtained, a limestone or mixture of the following composition is preferred :

Magnesia	36	per cent.
Lime	52	"
Silica	8	"
Alumina and oxide of iron	4	"
	<u>100</u>	

For the above analysis of what is now commonly known as 'basic' lining I am indebted to Mr. Percy Gilchrist, one of the patentees of the process.

It is being attempted to produce magnesia on a large

scale by precipitating it from its chloride with milk of lime. It is simply a question of cost. If it can be done economically a very useful substance for the manufacture of basic bricks will be available. Magnesia exists in considerable quantity in several waste liquors.

If what has been said on the subject of materials capable of resisting the action of fire has been clearly understood, it will have become evident that, unless previously fired they should only be used in such positions as will ensure their being subjected to a high temperature. It would, therefore, be objectionable to employ fire-clay mortar in a cool part of a furnace, as it would inevitably crumble out. On the other hand, ordinary mortar, which consists of sand and lime, is not suitable for use, except in the cool parts of a furnace.

THE METALS.

THE word METAL appears to be derived from the Greek μετ' ἄλλα, *in quest of other things*, whence come μεταλλάω, *to search after, to explore*, or, in gold-diggers' language, *to prospect*, and the corresponding substantive μέταλλον, *a mine*.

About fifty of the undecomposed or *elementary* substances are classed together under the head of METALS by the chemist, because they manifest certain properties when acted upon by chemical tests, without regard to those external characters which are commonly associated with the idea of a metal.

Many of these are unfit to be employed in the metallic state for any of the ordinary uses of metals, because they cannot be exposed to the action of air, even for a short time, without being rusted or corroded, by combining with the oxygen of the air, to such an extent that they entirely lose their metallic characters.

Among those which offer sufficient resistance to the action of air, many are excluded from useful application in their metallic state, on account of their rarity, or of the great difficulty which is experienced in extracting them from their ores.

The metals which are employed for useful purposes in their pure or metallic state are—

Aluminum	Copper	Magnesium	Platinum
Antimony	Gold	Mercury	Silver
Bismuth	Iron	Nickel	Tin
Cadmium	Lead	Palladium	Zinc.

On considering this list, it will be seen that several of the metals named in it are employed to produce some effect dependent upon a peculiar property of the metal, and not upon qualities which belong to it in common with the rest. Thus, mercury or quicksilver is used for *amalgamating* or dissolving other metals, and also as a suitable liquid for constructing barometers and thermometers; antimony owes its usefulness to its property of hardening lead and tin when melted with them; bismuth and cadmium are employed to render lead and tin capable of being melted at lower temperatures; nickel is used to whiten copper in order to make German silver; and magnesium is valuable for its property of burning easily with production of a brilliant white light.

Moreover, gold, platinum, palladium, and silver, being comparatively rare, and aluminum being obtainable by a somewhat costly process, the useful applications of these metals are limited by their high price, so that there remain only TIN, LEAD, COPPER, IRON and ZINC to be considered as metals largely employed for useful purposes.

The qualities possessed by these metals, rendering them fit for purposes which could not be fulfilled by non-metallic substances, are *lustre*, or the power of reflecting light; *tenacity*, or resistance to any attempt to pull asunder their particles; *malleability*, or the capability of being hammered or rolled into thin sheets; *ductility*, or the property of being drawn out into wire; high *specific gravity*, or relative weight; high *conducting power* for heat and electricity; and *fusibility*, or the property of becoming liquid when heated.

METALLIC LUSTRE.—The power of reflecting the rays of light is possessed in a much higher degree by metals than by non-metallic substances. Although some examples of the latter class, such as iodine and plumbago, reflect much of the light which falls upon smooth surfaces of them, they have a black appearance, caused by their absorbing a large proportion of the luminous rays, which is quite different from the true metallic lustre. Iron, in the form of steel, is

capable of exhibiting this lustre in very great perfection, because the hardness of steel allows its surface to be ground perfectly smooth by the application of fine particles of very hard substances, such as emery and diamond-dust, which rub off minute projections from the surface without producing scratches or indentations. A surface so polished sends back directly to the eye of the observer almost all the light falling upon it, whilst a rough surface, being made up of a number of small surfaces, scatters the reflected rays in all directions. Tin is naturally a brilliant metal, but is not hard enough to be polished, like steel; if, however, it be dissolved in twice its weight of melted copper, an *alloy* of great hardness and brilliancy is formed, which is employed for the *specula* or mirrors of reflecting telescopes. Zinc and lead exhibit the metallic lustre in an inferior degree, and become dull when exposed to air, because the metal at the surface combines with oxygen, forming a thin film of oxide which has no metallic lustre. The natural lustre of silver is very great, and, if it be hardened by admixture with a little copper, it becomes susceptible of a very high polish which is not dimmed by the action of the oxygen of air, though it is easily tarnished by sulphur existing in foul air in the form of sulphuretted hydrogen. The splendid combination of lustre and colour exhibited by burnished gold is proverbial, and is undiminished by the action of the atmosphere. The lustre of palladium and platinum resembles that of silver, and is not affected by oxygen or sulphur in the air. Aluminum has also a permanent lustre, though inferior to that of silver. When dissolved in nine times its weight of melted copper, aluminum forms a hard yellow alloy capable of being polished to resemble gold, but becoming slowly tarnished by the action of the oxygen in air.

TENACITY.—The strength with which the metals oppose any attempt to pull asunder their particles is one of their most useful properties, and is determined by ascertaining the exact weight which must be suspended from the ends of

wires or rods of equal diameter, in order to break them. The weight required to break a given metallic wire is found to vary according to the manner in which the strain is applied, the resistance of the wire being greater when the whole of the breaking weight is applied at once than when it is added gradually, probably because, in the latter case, the wire becomes stretched and weakened by each additional weight.

Steel (iron combined with about $\frac{1}{100}$ th part of carbon) is by far the most tenacious of metals, and lead is the least tenacious of those in ordinary use.

If the weight required to pull asunder a wire of lead be taken as unity, that required by similar wires of the other metals will be found to approach nearly to the numbers contained in the following table:—

Relative Tenacity of the Metals.

Lead	1	Silver	$12\frac{1}{2}$
Tin	$1\frac{1}{3}$	Platinum	15
Zinc	2	Copper	18
Palladium	$11\frac{1}{2}$	Iron	$27\frac{1}{2}$
Gold	12	Steel	42

The tenacity of metals is very seriously affected by variations in their structure, purity and temperature. Thus, rods of metal which have been cast in a mould are generally weaker than rods of equal dimensions made by drawing the metal through the gradually diminishing holes of the wire-drawer's plate. The tenacity of iron rods which have been rolled until they have acquired a fibrous structure, is much higher than that of rods which are *crystalline* in texture, the metal tending to break asunder where the smooth surfaces of the separate crystals are in contact with each other.

The tenacity of a metal when hot is, as might be anticipated, less than its tenacity when cold; and if the metal be made red-hot and allowed to cool slowly, it will generally be found to have diminished in tenacity, probably because

a high temperature tends to encourage the formation of a crystalline structure. The effect of the presence of impurities upon the tenacity of metals will be more appropriately studied when the individual metals are under consideration, but it may be stated generally that chemical purity is not of necessity accompanied by the highest degree of tenacity. Thus, the small proportion of carbon present in steel is seen in the above table to have greatly increased the tenacity of the iron, and pure zinc has a much lower tenacity than the ordinary zinc of commerce.

MALLEABILITY.—The facility with which a mass of metal can be hammered or rolled into a thin sheet without being torn, must depend partly upon its softness, and partly upon its tenacity. If it depended upon softness alone, lead should be the most malleable of ordinary metals; but, although it is easy to hammer a mass of lead into a flat plate, or to squeeze it between rollers, any attempt to reduce it to an extremely thin sheet fails from its want of tenacity, which causes it to be worn into holes by percussion or friction. On the other hand, if malleability were entirely regulated by tenacity, iron would occupy the first place, whereas, on account of its hardness, it is the least malleable of metals in ordinary use; whilst gold, occupying an intermediate position with respect to tenacity, is the most malleable, which appears surprising to those who are only acquainted with gold in its ordinary forms of coin and ornament, in which it is hardened and rendered much less malleable by the presence of copper and silver.

During the rolling or *lamination* of metals their particles are obviously squeezed into unnatural positions; it becomes necessary, therefore, in order to avoid breaking, to enable the particles to resume their former relative situations; this is effected by heating the metallic sheet after every two or three rollings, and allowing it to cool slowly, a process of *annealing* similar to that by which glass vessels are rendered less brittle.

In the following table the ordinary metals are arranged in the order of malleability :—

Table of Malleability.

1. Gold.	4. Tin.	7. Zinc.
2. Silver.	5. Platinum.	8. Iron.
3. Copper.	6. Lead.	

DUCTILITY.—The ease with which a metal can be elongated into a wire, by being drawn through the gradually diminishing holes of the wire-drawer's plate, will be greater in proportion to the softness of the metal; but the thinness of the wire to which it can be reduced is regulated by the tenacity of the metal, which enables it to resist, without breaking, the force required to draw it through the holes. And it is found that their tenacity has more influence upon the ductility of metals than upon their malleability, for the particles of a weak metal, like tin, may cohere under the hammer, although they would be easily torn apart by the direct pull necessary in wire-drawing.

Gold, silver, and platinum, which occupy an intermediate position with respect to tenacity, are the most ductile of the metals, whilst tin and lead, which are lowest in tenacity, are the least ductile, though their softness gives them a higher place in the order of malleability.

Table of Ductility.

1. Gold.	5. Copper.	8. Zinc.
2. Silver.	6. Palladium.	9. Tin.
3. Platinum.	7. Aluminum.	10. Lead.
4. Iron.		

The metals require annealing during the process of wire-drawing, as in that of lamination, and for a similar reason.

SPECIFIC GRAVITY.—The relative weights of equal bulks of the metals exercise considerable influence upon their useful applications. The relative weight of gold being very high it is well adapted for a circulating medium, a large

value being compressed into a portable form. On the other hand, iron would be employed with far less advantage in building if its relative weight did not happen to be low, whilst aluminum, being the lightest of metals in ordinary use, is particularly well adapted for the production of small weights, as fractions of a grain, which shall yet be large enough to handle; such weights being nearly nine times as large when made of aluminum as they are when platinum is employed, as was the case before the introduction of aluminum.

The *specific gravities*, or comparative weights of equal bulks of the metals, are generally expressed by numbers which show that each metal is so many times as heavy as an equal bulk of pure distilled water at the ordinary temperature (60° F.); thus, zinc is a little more than seven times as heavy as an equal bulk of water, so that its specific gravity is expressed by 7 and a fraction.

The first column of numbers in the following table gives the specific gravities of the metals in round numbers, which can be easily retained in the memory, and are sufficiently exact for ordinary purposes, the more accurate numbers usually employed in scientific works being given in the next column :—

Table of Specific Gravities of the Metals.

Platinum	. 21 $\frac{1}{2}$. 21.53	Nickel .	. 8 $\frac{1}{2}$. 8.82
Gold .	. 19 $\frac{1}{3}$. 19.34	Iron .	. 7 $\frac{1}{2}$. 7.84
Mercury	. 13 $\frac{1}{2}$. 13.59	Tin .	. 7 $\frac{1}{2}$. 7.29
Palladium	. 11 $\frac{1}{2}$. 11.8	Zinc .	. 7 $\frac{1}{7}$. 7.14
Lead .	. 11 $\frac{1}{4}$. 11.36	Antimony	. 6 $\frac{1}{2}$. 6.71
Silver .	. 10 $\frac{1}{2}$. 10.53	Aluminum	. 2 $\frac{1}{3}$. 2.67
Bismuth	. 9 $\frac{1}{2}$. 9.79	Magnesium	. 1 $\frac{3}{4}$. 1.74
Copper	. 9	. 8.95			

CONDUCTING POWER OF METALS FOR HEAT.—The sensation of cold when the hand is placed upon a piece of metal of the ordinary temperature of the air shows us that metals are better conductors of heat than non-metallic

bodies, for the particles of metal which are first warmed by contact with the hand give up the acquired heat to the neighbouring particles, and being thus cooled to nearly their former temperature, are able to abstract a fresh supply of heat from the hand ; whereas, when the hand is placed upon wood, or other inferior conductors of heat, the particles in contact with it are warmed by the removal of a trifling amount of heat from the hand, and are not soon cooled again by parting with their heat to the particles adjoining. In consequence of the rapidity with which heat applied to one portion of a mass of metal is communicated to the whole of the particles composing it, metals may be suddenly heated or cooled with much less risk of causing them to crack or fly than is the case with non-metallic substances. When an earthenware pipkin or a glass bottle is placed upon the fire, the outside immediately becomes much hotter than the inside, and being expanded by the heat, tears apart the particles of the inside of the vessel and produces a crack, but in the case of a metallic vessel the heat is rapidly transmitted, and all parts of the vessel are expanded almost simultaneously. The much greater rapidity with which water can be heated in metallic vessels is another useful result of the superior conducting power of the metals.

In the following table the metals are arranged in the order of their conducting power, the first being the best conductor :—

Table of Conducting Power for Heat.

1. Silver.	5. Zinc.	9. Lead.
2. Gold.	6. Iron.	10. Antimony.
3. Copper.	7. Tin.	11. Bismuth.
4. Aluminum.	8. Platinum.	

CONDUCTING POWER OF METALS FOR ELECTRICITY.—

The conducting power for electricity, of metals, refers to the facility with which an electric disturbance excited in one portion of a mass of metal is transmitted to the other particles composing the mass. Thus, a very slight electric dis-

turbance at one end of a copper wire is sufficient to produce movement in a telegraph needle at the other extremity, whilst a much greater amount of disturbance, or, in other words, a more powerful *current*, is required if an iron wire of the same length and thickness be employed.

Only one non-metallic substance—carbon, in some of its varieties—at all approaches to the metals in the power of conducting electricity.

Those metals which are the best conductors of heat are also the best conductors of electricity, and in both cases the conducting power is seriously impaired by the presence in the metal even of small quantities of other metals, or of non-metallic bodies, as well as by an increase of temperature in the metal. When heated to the boiling point of water, the metals have only about three-fourths of the conducting power which they exhibit at the freezing point.

The following table shows the relative conducting power of the most important metals, in a pure state, at 32° F., the conducting power of silver, which is higher than that of any other metal, being taken as 1000:—

Table of Conducting Power for Electricity.

Silver = 1000

Copper . . .	999	Nickel . . .	131
Gold . . .	779	Tin . . .	123
Zinc . . .	290	Lead . . .	83
Palladium . . .	184	Antimony . . .	46
Platinum . . .	180	Bismuth . . .	12
Iron . . .	168		

FUSIBILITY.—Although the property of becoming liquid at high temperatures is not confined to the metals, it must be mentioned among the properties which conduce to their utility, for it enables the founder to produce a large number of objects of a given pattern with little expenditure of time and labour, and offers to the worker in metals a ready method of soldering together, in a durable manner, the sepa

rate pieces of his work. Tin and lead, being the most fusible of ordinary metals, are the constituents of solder, whilst iron (wrought iron), as the least fusible of the common metals, is used for firebars, melting-pots, and similar purposes.

Table of Fusibility.

Tin	melts at	442° F.	Silver	melts at	1800° F.
Cadmium	„	442	Copper	„	1990
Bismuth	„	507	Gold	„	2000
Lead	„	617	Cast Iron	„	2780
Zinc	„	773	Steel	„	4000
Antimony *	„	1150	Wrought Iron	„	above 4000

Platinum melts only in the oxy-hydrogen blowpipe flame.

In practical work the temperature is commonly inferred from the appearance of the fire ; thus, the *red heat* of an ordinary domestic fire is roughly valued at 1000° F., so that tin, lead, and zinc can be very easily melted in a crucible or ladle placed in such a fire ; but aluminum, silver, copper, and gold, require a *bright red (cherry red)* or furnace heat to melt them ; cast iron requires a very bright red heat, only attainable in a furnace with a very good draught ; and for melting steel, a furnace of special construction (*wind furnace*) is employed. Wrought iron can be fused only at a white heat, producible by a blast of air in a forge, and platinum melts at a greenish white heat in the flame of hydrogen supplied with pure oxygen. The production of a temperature adequate to the fusion of steel and wrought iron in large quantities has been much facilitated by the introduction of Siemens' regenerative furnace, in which the waste heat of the fire, instead of escaping up the chimney into the air, is accumulated in masses of fire-brick, and restored again to the furnace.

* Estimates of temperature above the fusing point of zinc cannot be regarded as exact, on account of the difficulty of ascertaining them.

IRON.

With respect to its useful properties, iron occupies the first place among the metals. By far the strongest, and, at the same time, one of the lightest, its applications in the arts of construction are much more numerous than those of any other metal. Being capable of assuming, according to the treatment which it undergoes, the forms of malleable iron, cast iron, and steel, it is susceptible of the widest variations in its characters. Extracted from its ores in the form of cast iron, it is melted with comparative facility, and, according to the mode of operating in the foundry, may be made to yield castings which are easily filed and turned, or may be rendered so hard that no tool is able to touch it. By judicious treatment with heat and atmospheric air the cast iron is converted into steel, the strongest, and one of the hardest and most elastic of all materials, as well as the only one of which a magnetic needle can be made. Continued a little further, the joint action of heat and atmospheric air converts the steel into malleable iron, possessing great strength and toughness, yet soft enough to be turned, bored, and punched with ease; and, especially when heated, to be easily wrought into the most varied forms without cracking. With less disposition to melt under the action of heat than any other common metal, wrought iron is sufficiently softened at a bright red heat to be *welded* or joined to another piece in the most perfect manner, without the use of solder of any kind. Being capable of acquiring and of losing the properties of a magnet with great rapidity, soft iron (wrought iron) is the only material which is adapted for the construction of electro-magnetic and magneto-electric apparatus.

It is not too much to assert that scarcely a step of importance has ever been made in the industrial progress of

any community to which some one of the three modifications of iron has not been indispensable.

Possessed of so many valuable qualities, iron is still the cheapest of all the metals, since the ores from which it is extracted are scattered in profusion through the crust of the earth, and can be made to yield the metal in abundance by a moderate expenditure of time, labour, and fuel.

ORES OF IRON.—Iron in the metallic condition, or *native iron*, is very rarely found in nature. Nearly all the specimens which have been examined have been *meteoric iron*, occurring in masses of irregular form, which have descended upon the surface of the earth, but whence they are derived is at present only a matter for speculation. Such masses have been found containing 93 parts in the hundred, of metallic iron, always associated with nickel, and sometimes with small quantities of other metals, and of phosphorus, sulphur and carbon. They vary much in size; two masses of iron, supposed to be of meteoric origin, have been recently found on the coast of Greenland, weighing, respectively, 21 tons and 9 tons. A small one, which was found at Lenarto in Hungary and weighed about 190 lbs., was remarkably malleable, and its analysis furnished the following results:—

Lenarto Meteoric Iron.

Specific Gravity, 7.79

Iron	.	.	.	90.883
Nickel	.	.	.	8.450
Cobalt	.	.	.	0.665
Copper	.	.	.	0.002
				<hr/>
				100.000

A recent examination of this meteoric iron has led to the very interesting discovery that it contains about twice and a half its volume of hydrogen gas, apparently in an uncombined state.

Iron is most commonly found in a state of chemical combination with oxygen or sulphur, which disguise its metallic

properties and convert it into earthy or stony masses. The compound of iron with oxygen, or *oxide of iron*, which is familiar to us in the form of *rust*, occurs in a very large number of mineral substances, and is often the cause of their colour. Sand, clay, and gravel, commonly owe their yellow, brown, or red shade to the presence of oxide of iron, a small proportion of which imparts a very distinct colour. No mineral substance, however, would be considered as an *ore* of iron which contained less than about twenty parts of iron in the hundred, for otherwise it would not repay the cost of its extraction.

The following table includes the mineral substances which are commonly regarded as ores of iron :—

Ores of Iron.

	Composition	Iron in 100 parts of pure Ore
Magnetic Iron Ore .	Iron, Oxygen	. 72
Red Hæmatite .	Iron, Oxygen	. 70
Specular Ore .	Iron, Oxygen	. 70
Brown Hæmatite .	{ Iron, Oxygen, Water }	. 60
Spathic Iron Ore .	{ Iron, Oxygen, Carbonic Acid }	. 48
Clay Iron Stone .	{ Iron, Oxygen, Carbonic Acid, Clay }	. Variable, 17 to 50
Black-band Ore .	{ Iron, Oxygen, Carbonic Acid, Clay, Bitumin- ous matter }	. Variable, 21 to 43
Iron Pyrites* .	Iron, Sulphur	. 46

Magnetic Iron Ore, or *Magnetite*, derives its name from Magnesia in Asia Minor, where its power of attracting iron and steel was first observed. One variety of the ore constitutes the *loadstone*,† which confers magnetic properties upon

* Iron Pyrites is not worked as an ore of iron, on account of the great difficulty in separating all the sulphur from the metal.

† Probably corrupted from *lead-stone*, the stone which leads or guides, in allusion to its use for making the needle of the mariner's compass

steel. This variety occurs chiefly in Siberia and the Hartz. Some of the common varieties of the ore do not attract iron, although they are capable, like steel, of becoming magnetic when brought into contact with powerful magnets.

This ore is generally met with in compact heavy masses of an iron black or grey colour, and with considerable lustre. Its specific gravity varies from 4.9 to 5.2. It abounds chiefly in the northern parts of the globe, and is found in immense masses in Norway, Sweden, Russia, and North America, being the most important iron ore in those countries. The iron extracted from it is generally of excellent quality, though it is occasionally deteriorated by sulphur and phosphorus, which are derived respectively from iron pyrites and from *apatite* (phosphate of lime), which are sometimes found associated with the magnetic ore. The bulk of the Swedish iron, so much valued for the manufacture of steel, is extracted from the magnetic ore at Dannemora, where it is worked in an open quarry.

Magnetic iron ore often contains titanitic acid, the oxide of the metal titanium, and this is especially the case with a variety of the ore which occurs as a heavy black shining sand in India, Nova Scotia, and New Zealand.

Red Hæmatite has been so called from the Greek word signifying *blood*, on account of its dark red colour,* and is sometimes erroneously called *bloodstone* (the true bloodstone being a dark green variety of silica (*heliotrope*) with red spots). In appearance it is the most striking of the ores of iron, sometimes appearing in rounded masses having externally a liver colour with considerable lustre, and internally made up of layers having the appearance of the thick shell of some huge fruit, or of bundles of fibres, which look like petrified wood. The specific gravity of this variety is

magnetic. In a similar way we have *load-star*, the star which leads towards the pole; *loadsman*, one who guides—a pilot; *load* or *lode*, the leading vein in a mine.

* The termination *ite*, so generally found in the names of minerals, was originally derived from the Greek word for *stone*.

about 5·0. Such specimens are in general remarkably hard, and are useful for burnishing metals. But it is occasionally found in much softer earthy-looking masses of a brighter colour, and not uncommonly associated with clay.

Red hæmatite is much more generally diffused than magnetite, and is found abundantly in this country, at Whitehaven (Cumberland) and Ulverstone (Lancashire), as well as in Glamorganshire. The compact variety is exceedingly pure, and furnishes iron of the very best quality. For smelting purposes any ore consisting essentially of anhydrous peroxide of iron is termed red hæmatite. Large quantities of uncrystallised earthy hæmatite, very rich in iron, are now obtained from Spain and Algeria.

This ore is also abundant in Ireland (Balcarry Bay), North America, Saxony, Bohemia, and the Hartz.

Specular Iron Ore is identical in composition with red hæmatite, though its appearance is so different, for it forms crystalline masses, and sometimes separate crystals of a steel-grey colour and brilliant lustre, whence it derives its name (*speculum* [Latin], a mirror). It is also called *Iron-glance* (*Glanz* [German], lustre), and in one of its varieties, made up of shining scales, *Micaceous Iron Ore* (*micare* [Latin], to shine). When scratched with a knife, or reduced to powder, it exhibits the red colour of hæmatite. The specific gravity of specular iron ore is 5·2. The island of Elba has long been noted for its specular iron ore, and it is also found in considerable quantity in Russia and Spain. The excellent iron of Nova Scotia is extracted from the specular ore of the Acadian mines.

Brown Hæmatite contains the same oxide of iron as red hæmatite, but in a state of chemical combination with water, the latter varying from ten to fourteen parts in a hundred parts of the ore. Its appearance varies widely in different specimens. Some form globular masses of considerable size, others occur in small round grains (*pea iron ore*); occasionally it occurs in stalactites; the various soft earthy

*ochres** and *umbers*† consist, for the most part, of brown hæmatite, the dark brown shade, in the latter case, being caused by the presence of an oxide of manganese.

In this country brown hæmatite is found at Alston Moor (Cumberland) and in Durham, but it is a much more important iron ore in France. Some varieties of it contain a considerable proportion of phosphorus (in the form of phosphoric acid combined with oxide of iron), which materially affects the quality of the iron extracted from them.

The *Black Brush Ore* of the Forest of Dean is a brown hæmatite containing 89 parts of peroxide of iron, and 10 parts of water, and yields most of the iron used for making tin-plate.

The *Lake Ores* of Sweden consist of a variety of brown hæmatite ore which occurs at the bottom of lakes, and is collected by dredging with a kind of iron sieve attached to a long pole, and thrust down through a hole in the ice, the mud being washed away from the ore by shaking the sieve up and down under water. In this way a man will sometimes collect as much as a ton of ore in a day.

Spathic Iron Ore (*Spath* [German], spar), or *sparry iron ore*, is composed of carbonate of iron, or iron combined with oxygen and carbonic acid. Some specimens consist of a collection of nearly transparent shining crystals, which are almost colourless, and have the same crystalline form as calcareous spar (carbonate of lime). It is found in extensive beds in Styria and Carinthia. Spathic ore almost invariably contains manganese, which especially adapts it for the manufacture of certain kinds of steel, whence it has sometimes been termed *steel ore*. This ore is also found at Wearsdale in Durham, and on the Brendon Hills in Somersetshire.

Clay Iron Stone, *Argillaceous*‡ *Iron Ore* or *Clay-band*

* Derived from the Greek for *sallow*, in allusion to their yellow colour.

† From *umbra* (Latin), shade, on account of their darker tint.

‡ *Argilla* (Latin), clay.

contains the iron in the same form of chemical combination (carbonate of iron), in which it exists in spathic iron ore, but in a state of intimate mixture with clay. This ore is by far the most important of British iron ores, being that which is most extensively worked in this country. It occurs in great abundance around Dudley, in Staffordshire, in Yorkshire, Derbyshire, and in South Wales. Both this ore and the black-band ore are found in layers which occur alternately with beds of coal, limestone, clay, and shale, whence they are often spoken of as the *Ironstones of the Coal Measures*, and the circumstance that the same pit, or neighbouring pits, will furnish the coal employed for smelting the ore, the limestone used as a *flux*, and even the clay for making fire-bricks for the furnace, allows English iron to be produced at a price with which, until lately, other countries found it impossible to compete. The coal formations of Belgium and Silesia also furnish large supplies of clay iron-stone.

This ore is found sometimes in continuous beds, and sometimes in irregular globular masses, imbedded in clay; it is moderately hard and stony, and varies in colour, through different shades of grey, slaty-blue, and brown. It is lighter than the preceding ores, and a cursory observer might regard it as a stone rather than a metallic ore. The proportions of carbonate of iron and clay contained in the ore vary considerably, the former amounting to 80 or 90 parts in the hundred in some specimens, and in others not exceeding half the weight of the ore.

Blackband Ore differs from clay iron-stone only by containing, in addition to its other constituents, a quantity of bituminous or coaly matter, sometimes amounting to one-fifth of the weight of the ore, and imparting the colour from which it derives its name. Lanarkshire and Ayrshire, in Scotland, contain extensive deposits of blackband iron ore, first brought to light in 1801. The presence of so much combustible matter enables the ore to be calcined without

expenditure of fuel. Blackband ores are also mined in Prussia.

The Northamptonshire iron ore, which is found in the *oolite* limestone in that county, appears to have been formed by the chemical alteration of clay ironstone, under the influence of air and water, for it contains carbonate of iron associated with clay, and with the red oxide of iron, which is formed by the action of air upon the carbonate. Since this ore contains much phosphoric acid, it is unsuited for the manufacture of malleable iron by the *puddling* process.

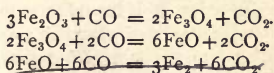
*Iron Pyrites** is the yellow substance, of metallic appearance, which is so common in lumps of coal, and may be found in rusty globular masses on the sea-beach. It is composed of $46\frac{1}{2}$ parts of iron in chemical combination with $53\frac{1}{2}$ parts of sulphur, and is extensively used as a source of sulphur in the manufacture of sulphuric acid. Attempts have been made to extract iron from this mineral after the sulphur has been burnt off, and the iron is left in combination with oxygen from the air, but they have not been attended with much success, owing to the fine state of division of the ore after calcination, and the expense of consolidating it by pressure.

EXTRACTION OF IRON FROM ITS ORES, IN THE FORM OF MALLEABLE IRON, STEEL, AND PIG OR CAST IRON.

Although it is commonly conceded that bronze was in general use before iron, yet there is undisputed evidence to show that malleable iron was known and employed at least 2,000 years ago. Nothing can be simpler than the extraction of iron from rich ores. All that is necessary is to heat them to redness with carbonaceous matter; say charcoal, when the oxygen of the ore will combine with the carbonic

* The word *pyrites* is derived from the Greek for *fire*, probably in allusion to the circumstance that when this mineral is heated it gives off sulphur, which takes fire.

oxide produced from the partial oxidation of the carbon, forming carbonic acid gas, which passes off into the atmosphere, and metallic iron, which is set free. However pure the ore may be it will always contain some earthy matter, with which the iron, on being *reduced* from its ore, will be mixed. In order to separate the iron from the earthy matter the latter must be rendered fluid, so that the iron, by reason of its greater specific gravity, may sink through it and collect together. One of the principle constituents of the earthy matter is usually silica, and it so happens that protoxide of iron is the very substance which, more readily than any other available, forms a fluid compound with silica at a low temperature. The ores employed by all primitive workers were peroxides, or if a protoxide (carbonate) were used it was first calcined, which would have the effect of driving off the carbonic acid, and converting the protoxide into peroxide by combination with a further quantity of oxygen. During the reduction of the ore by the carbonic oxide, the peroxide is first reduced to magnetic oxide, then to protoxide, and finally to metallic iron, thus :—



Some of the protoxide produced in contact with the earthy matter combines with it, rendering it fluid, and forming what is known as *slag*, a term applied to all earthy matter which has been separated in a more or less fused condition during the reduction of a metal from its ore. The result of heating an iron ore, consisting essentially of peroxide of iron, with carbon is, then, to produce a pasty mass of malleable iron, charged with more or less fluid slag, or *cinder*, as it is also termed. In order to remove the cinder and consolidate the iron, it only remains to subject the mass to sufficient pressure whilst still red-hot, when the slag will be squeezed out, and the metal owing to the

property of *welding*, which iron possesses in a high degree, will unite to form a solid mass, which can, subsequently, by reheating and hammering, be worked into any desired form.

By such simple means as these iron has been made in India from time immemorial, and it continues to be so made both there and in Africa at the present day. The most primitive method of all is that practised by the Burmese. The furnace consists simply of a hole about 10 feet deep, dug in a bank, at a distance of 2 or 3 feet from its edge. The face of the bank is strengthened by pieces of wood placed crosswise ; the lower part of the bank is dug away, and a rectangular opening, about 1 foot high and the whole breadth of the furnace, is made, through which the metal and slag are removed at the completion of the smelting operation. The opening at the base is closed with moist clay, through which are inserted about twenty small clay tubes, or *twyers*. These tubes are made by plastering moist clay over stems of wood, which are then cut into lengths of 4 inches, and burnt. The *twyers* thus made are placed in the opening, in a single line, about half-way up. Lighted wood is then thrown into the furnace, and on it alternate layers of charcoal and ore, until the furnace is charged to the top. After the lapse of about eight hours the slag is removed, through an opening temporarily made for the purpose ; this operation is repeated every half-hour, until no more slag comes away. The smelting is complete in about twenty-four hours ; the weight of the iron produced averages about 90 lbs.

In some parts of India an artificial blast is employed, produced by bellows made from goat skins, or by means of some primitive form of blowing-cylinder ; the furnaces also show an advance in constructive power as compared with that just described, as they are built above ground. In Central India and the North-Western Provinces, where manufactures have received the greatest development, *continuous working* is effected by removing the iron by means

of tongs through the top of the furnace, the slag being run out, or *tapped*, through a small opening at the base. In this way, the furnace not being damaged by the withdrawal of the lump of iron, it can be re-charged again whilst still hot ; thus time, fuel, and labour are saved.

In Spain a somewhat similar method of producing malleable iron direct from the ore has been in use, with but

slight modification, since the earliest times of which we have any record.

This process is commonly known as the *Catalan*, owing to its having been principally practised in the provinces of Catalonia, where the magnetic ore and the hæmatite of the Pyrenees are smelted with the charcoal made in the neighbouring forests. The smelting works com-

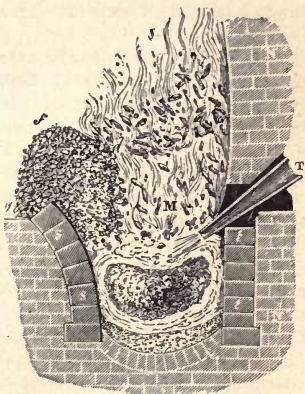


FIG. 18.—Catalan Forge.

prise a forge, a blowing-machine, and a hammer, but the first alone will be here described in order to illustrate this method of treating iron ores.

The crucible or hearth is a nearly rectangular trough (M, Fig. 18) well built around with masonry, about 17 inches deep, 21 inches long, and $18\frac{1}{2}$ inches wide. The bottom of the crucible is composed of a block of granite, which is supported upon small arches to keep it dry.

That side of the hearth at which the blast from the tuyère (T) enters is perpendicular, being built up of massive pieces

of iron (*t*), the blast-pipe, or tuyère, of copper, being supported upon the uppermost piece in such a manner that its inclination to the bottom of the crucible can be varied at pleasure, since this exercises much influence upon the success of the operation. The wall opposite to the blast is built up of wedge-shaped pieces of iron (*s*), and presents a curved surface. The working side of the hearth is composed of three thick pieces of iron placed end to end, the side opposite to it being lined with fire-clay, and having a moderate inclination.

Mode of Conducting the Process.—The ore is crushed by the hammer, and divided by sifting into lumps (*mine*) and very coarse powder (*greillade*). The furnace being still red-hot from the last operation, it is filled with charcoal nearly to the twyer; the hearth is then divided at a point about two-thirds distance from the twyer into two parts by a broad shovel; on the blast side a further quantity of charcoal is added, whilst that on the other side having been rammed down firm, ore is added so as to fill that part of the furnace. On this is placed moistened charcoal dust, except at the highest point; a good blast is then turned on, and, if the whole is in good order, jets of blue flame at once issue from the uncovered portion of the ore. After a few minutes the pressure of the blast is lowered to 1·5 inch of mercury. At intervals during the process—which lasts about six hours—the blast is gradually raised until it reaches about 3 inches, the maximum usually employed.

During the whole of the process, at short intervals, *greillade* and charcoal are added, and well moistened with water to prevent too rapid combustion. After about two hours from the commencement the wall of *mine* (*i.e.* ore in lumps) is pushed well forward under the twyer, and more *mine* is thrown into the space thus made. This part of the process is also subsequently repeated at intervals, until sufficient has been added to form a lump of iron, or *massé*, of the required size. From time to time slag is removed by opening the

tap-hole. At the completion of the process a mass of metal is obtained weighing about 3 cwt., which invariably consists partly of soft iron and partly of *steely* iron and *steel*, i.e. iron containing more or less carbon, for the conditions in this furnace are very far from being uniform.

Reactions in the Furnace.—We have seen that in the one part of the furnace only charcoal and *greillade* are introduced, and in the other only lumps of ore. That the ore should be in lumps at that part is a very important point, for in this way the hot reducing gas, carbonic oxide (CO), generated by the action of the blast on the charcoal, is able to pass freely through the mass of the ore, the effect of which is that the water of hydration and the moisture are first driven out by the heat, and then, the ore having become easily permeable, the carbonic oxide reduces it to metallic iron by successive stages, as already described.

At the same time that these reactions are going on the ore has become impregnated with carbon, derived from the decomposition of the gases with which it is charged. That this would be the case, the experiments of Mr. Lowthian Bell and others can leave no manner of doubt.

On the twyer side, where are placed the charcoal and *greillade*, the latter, as the charcoal is burnt away, descends rapidly, and to a considerable extent, doubtless, escapes reduction; for the arrangement of the blast is such that most of the reducing gas is projected on to the lumps of ore, and does not pass up through that portion of the furnace occupied by the charcoal and *greillade*, which, besides, are constantly damped. The *greillade* is much richer in silica than the larger pieces, from which it results that the quantity of slag will vary with the *greillade* added, which is always very rich in oxide of iron.

What happens in this process appears to be this: carburised iron is produced by the gradual reduction and fusion of the lumps of ore, and this, coming in contact at the bottom of the furnace with slag, very rich in oxide of

iron, the carbon of the one combines with the oxygen of the other, and the result is that iron containing more or less carbon is produced, according as little or much oxide was present.

The obvious conclusion would be, that the less there was of *greillade* present, the more steely would be the iron: in practice this is found to be the case. This circumstance would naturally suggest the total suppression of the *greillade* when it was desired to produce steel. That would, however, be impracticable; for it is necessary that some of the oxide of iron should remain unreduced, in order to flux off the silica, which occurs in considerable quantity in the ore. In the blast-furnace this difficulty is got over by employing lime; but lime, at the temperature of the Catalan furnace, would not produce a sufficiently liquid slag.

All that can be done, then, when it is desired to produce steel, is to employ every available means to prevent decarburisation. In addition to using less *greillade*, the slag is tapped out more frequently, so that the lump of iron as it forms may remain as little time as possible in contact with it. The bank of ore is exposed for a longer time to the reducing and carburising gases, and is pushed more gently towards the twyer, so as not to become decarburised by the oxygen of the air, which has not had time to combine with the carbon of the charcoal. Lastly, manganese should be present. It is found that the presence of manganese has a very important influence, which is probably due to its power to replace iron in the slag. A slag in which some of the iron is replaced by an equivalent quantity of manganese is more liquid, and, according to François, has not the same tendency to cause decarburisation at the temperature of this process.

In order, then, that steel may be produced by this process, every precaution is taken to cause as much carburisation as possible, the unavoidable presence of oxide of iron in the slag and the low temperature effectually preventing the

formation of cast iron, a more highly carburised metal than steel ; the presence of oxide of iron, indeed, making it very difficult, as we have seen, to obtain steel.

It might be said: Why not increase the temperature, so as to obtain a liquid slag without using oxide of iron? If the temperature were increased, cast iron instead of steel would be produced ; in fact, that is exactly how cast iron first came to be obtained in blast-furnaces.

This process has been rather fully treated of, because the principle of it is not always well understood. Rightly looked at, it explains how steel was first obtained.

In the Indian furnaces steely iron and steel are also always produced in more or less quantity. In some districts the natives so well understand the necessary conditions that they can produce steel at will in their furnaces.

No matter whether it is the intention to produce malleable iron or steel, it is very difficult to obtain a homogeneous product from these furnaces. In practice the mass is broken up, and the pieces sorted according to their relative hardness and suitability for different purposes.

The circumstance that when a high temperature existed in the furnace, and much charcoal was present, the iron obtained was more or less steely, we have seen, was taken advantage of in India to produce more or less steely iron, by regulating the conditions in the furnace. Once these facts had been grasped, it required no great powers of reasoning to suggest the production of steel by heating malleable iron in contact with carbonaceous matter. The steel produced in this way by the Hindoos is known as *wootz* ; its manufacture dates from a very remote period, and is still carried on at the present day. They place in unbaked clay crucibles, of the capacity of a pint, a piece of malleable iron, some chopped wood, and a few leaves of certain plants ; the top of the crucible is then closed with clay, and the whole well dried near a fire. A number of these crucibles are then strongly heated for about four hours in a cavity in the

ground, by means of charcoal, and a blast of air forced in by bellows. There is some reason to believe that an excess of carbon over that required to produce the hardest steel has to be added, in order to fuse the metal at the temperature which can be commanded in these furnaces. Before being drawn out into bars the cakes of metal obtained in this way are exposed in a charcoal fire, during *several hours*, to a temperature a little below the melting-point, the blast of air playing upon them during the time. The object of this is, doubtless, to remove the excess of carbon by a process of *cementation*, which will be explained presently.

In 1800 a patent was taken out by David Mushet for a process in every respect analogous to that just referred to. He appears, however, to have applied it to the manufacture of a metal low in carbon, and therefore intermediate between iron and steel, partaking in a certain degree of the properties of both; corresponding, in fact, to what we have referred to as *steely iron*. Since this metal must have been in a state of fusion, Mushet must have brought to bear upon it a very high temperature. The manufacture was conducted in crucibles.

Malleable iron left long in contact with strongly-heated carbon, and protected from the direct action of the blast, becomes *steely*, and, if sufficient time be allowed, it is converted into steel without undergoing fusion or even becoming pasty. This circumstance could not long have escaped notice, and doubtless it was to the observation of this phenomenon that the method of producing steel known as the *cementation* process owes its origin. When or where the cementation process first came into use is not known. For many years past all the best steel for cutlery, &c. has been made in this way.

The cementation furnace (Fig. 19) is dome-shaped, like the furnace of a glass-house, and is enclosed in a conical jacket of brickwork, which serves to carry off the smoke from the flues. The hearth of the furnace is divided into

two parts by the grate (G), traversing the whole length (13 or 14 feet) of the furnace, in which a coal fire is maintained, the flame of which is made to circulate above, below, and around the fire-clay chests or *pots*, or *troughs* (C), placed one on each side of the grate, before escaping through the flues in the wall (H) and through the opening (M). These troughs are 10 or 12 feet long, and about 3 feet in depth and width, so that each will contain seven or eight tons of bar-iron, together with the charcoal necessary for its conversion into steel. A small opening is left at about the middle of one end of each chest, through which the end of one of the bars undergoing cementation is allowed to project ; this *proof-bar*

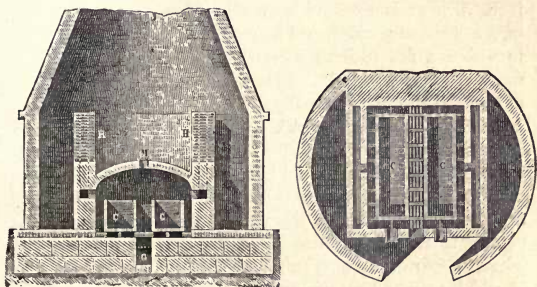


FIG. 19.—Cementation Furnace for converting Bar Iron into Steel.

is withdrawn from time to time, through a small door in the wall of the furnace, for the purpose of watching the progress of the cementation. There is also a small door in the wall of the furnace, a little above the top of each trough, through which the bars of iron may be introduced and withdrawn, a larger door being made in the middle of the wall to allow the passage of the workmen.

The cementing material is charcoal, in pieces from about a quarter to half an inch in diameter. Alkaline salts, and substances containing nitrogen are sometimes added, because

they cause the formation of cyanogen, which is said to assist the acquisition of carbon by the steel. The bars of iron should be of the purest description, if the best steel is to be produced. They are about three inches broad, and one-third of an inch thick.

In order to fill the troughs, the workman stands upon an iron platform between the two, and sifts the cement powder into them, so as to form a layer of about half an inch in depth, upon which the bars are arranged, standing upon their edges, at about an inch apart. More cement powder is now sifted over these, so as to fill up the intervals between them, and to cover them entirely to the depth of about an inch. Upon this a second layer of bars is placed, then more of the charcoal powder, and so on, until the trough is filled to within a few inches; it is then covered with grinders' waste—'wheel swarf,' consisting mainly of silica and oxide of iron. The fire is gradually applied during the first two or three days, to avoid the risk of splitting the troughs. A temperature high enough to melt copper (estimated at about 2,000° of Fahrenheit's scale) is required to enable the bar iron to acquire a proper proportion of carbon, and the troughs are maintained at this temperature for a period proportionate to the hardness which the steel is required to possess; four days being sufficient for producing the steel of which saws and springs are made, while six or eight days are required for shear steel, and ten days or more are required for the very hard steel of which cold chisels are made. The fire is then gradually let down, to avoid sudden change of temperature, so that some days elapse before the troughs are cool enough to be opened. About three weeks are commonly occupied in the conversion of the bar-iron into steel: one to get up the heat, one to keep it at the required degree, and one to cool it down; so that only about sixteen cementations can be executed in a year by a single furnace.

The bars are found to have upon their surface bubbles or blisters of considerable size, whence they are called *blister*

steel. On breaking the bars the fracture exhibits a silvery lustre and a well-marked crystalline structure. The proportion of carbon which has entered into combination with the iron depends upon the duration of the cementing process, but it rarely exceeds fourteen parts in a thousand parts of the metal.

The chemical changes which are involved in the process of cementation are not yet thoroughly understood. The passage of infusible solid carbon into the interior of the solid iron bar obviously requires explanation. It might be imagined that the external particles of iron which are in contact with the charcoal, becoming charged with carbon, impart a portion of that element to the next layer, and so on, until the particles in the very centre of the bar had acquired a share of carbon.

The following explanation is that usually accepted. The small quantity of oxygen contained in the air remaining in the trough, and present in the pores of the charcoal, enters into combination with the carbon to form carbonic oxide gas; this gas, in contact with iron at a high temperature, gives up one-half of its carbon to the metal, and becomes converted into carbonic acid gas; but this carbonic acid, in contact with the strongly-heated carbon, is reconverted into carbonic oxide, which again transfers one-half of its carbon to the metal, these changes recurring many times in the same order, until the whole of the iron is converted into steel. The observations of chemists during the last few years have shown that red-hot iron allows the passage of gas through its substance, and that this metal has the power of absorbing a considerable quantity of carbonic oxide, which renders it easy to account for the transference of carbon from the charcoal into the interior of the bar.

That part of the explanation which refers to the method by which the iron acquires carbon from the carbonic oxide is most assuredly erroneous. The molecule of carbonic oxide consists of one atom of carbon and one atom of

oxygen. Chemical theory requires that the atom should be the smallest particle of matter which can combine with other atoms. It is, therefore, manifestly absurd to talk of the iron taking half the carbon from the carbonic oxide, since the carbonic oxide contains but one atom, and atoms are indivisible.

A more probable explanation is, that the carbonic oxide occluded by the iron undergoes dissociation, depositing carbon and setting free oxygen, which immediately combines with a further quantity of carbonic oxide, forming carbonic acid. This point will be discussed more in detail further on.

Other gases containing carbon are capable of imparting that element to iron; thus, if coal gas, which contains carbon in combination with hydrogen, be passed for an hour through an iron tube containing some soft iron wires heated to bright redness, the wires will absorb carbon from the gas, and become converted into steel.

The blisters, which are distributed sparsely and irregularly over the surface of the bars, are now known to be due to the action of particles of oxide of iron, or of slag, accidentally occurring in the iron bars, upon the carbon combined with the iron, giving rise to carbonic oxide gas, the expansion of which causes the blister.

As might be anticipated, the blistered steel, in its present condition, is only fitted for very rough articles, such as shovels; its largely crystalline structure renders it deficient in tenacity, and the bars are further weakened by their want of uniformity and by the presence of the blisters.

Conversion of Blistered Steel into Tilted or Shear Steel.—The quality of the blister steel is improved by a process similar in principle to the fagotting of bar-iron. Five bars of blister steel are bound together into a bundle, being secured by a stout steel wire; four of the bars are about 18 inches long, and the fifth is twice that length, so that it projects beyond the others, and forms a handle. This bundle

is raised to a welding heat in a forge, sprinkled with sand to combine with the oxide of iron and to form a fusible slag, and is placed under the *tilt-hammer*. This hammer weighs about two hundredweight, and it is so suspended that it may be raised by *cams* projecting from the circumference of a wheel, the revolutions of which bring them down in succession upon the tail of the hammer, the head falling again upon the anvil as soon as the cam has passed. A few blows from this hammer soon weld the bars together, when the binding ring is knocked off, the bundle again heated in the forge, and hammered or *tilted* throughout its whole length, and on all sides, until it is reduced to a rectangular bar of the right dimensions. In order to avoid the necessity for reheating the bar during the process, the tilting must be effected with great celerity, and the hammer is made to deliver 300 or 400 blows per minute, the number being, of course, regulated by the rate at which the cam-wheel revolves. The workman, being seated upon a swinging bench which brings him upon a level with the anvil, is enabled to move to and fro with little effort, and to bring every part of the elongated bar under the strokes of the hammer. The fracture of a bar of shear steel shows it to possess a much more compact structure than the blister steel, and its tenacity and ductility have been much improved by the tilting. It is probable also that, as in forging bar-iron, the proportion of carbon has undergone a slight diminution, the steel being found to become softer after repeated tilting. If *double shear steel* be required, the tilted bar is broken, and the two pieces welded into a single bar.

Although the want of homogeneity in steel made as just described, owing to unequal carburisation, and to the presence of slag, which always exists in more or less quantity in wrought iron, was well known, yet nothing was done to remedy it until about the year 1760, when Huntsman conceived and carried into effect the idea of fusing the blister steel. He had great difficulties to contend with at first, for

there were no crucibles in existence then which would stand the requisite temperature. This was the first time steel was ever obtained in the molten state, unless we regard the Indian wootz as such, which, however, when in the molten state, was probably too highly carburised to constitute steel.

The fusion of blister steel was a great step forward. The molten metal is cast into ingots, and, after being carefully reheated, is drawn down under a tilt- or steam-hammer.

Conversion of Blister Steel into Cast Steel.—The blister steel is broken up into pieces of a convenient size for packing close together, and about 50 lbs. of it are introduced into a tall narrow crucible, about two feet high, made of fire-clay mixed with black-lead, and provided with a closely-fitting cover. Some steel-makers add a little bottle-glass, to

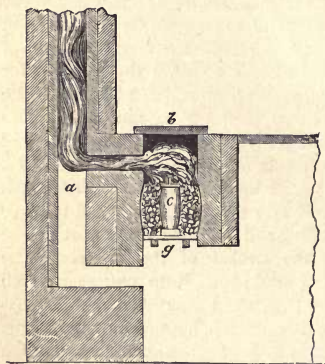


FIG. 20.—Furnace and Pot for melting Steel.
g, Grate. c, Crucible. b, Cover of Furnace.
a, Chimney.

fuse over the surface and prevent oxidation of the steel.

The crucibles are placed in a small furnace (Fig. 20) holding six, twelve, or more, about one foot wide and two feet deep, the opening of which is usually on a level with the floor, to facilitate the lifting of the crucibles. Several of these furnaces are connected by flues with the high chimney of the works, so that a powerful draught may be produced. Hard coke broken into small pieces is employed to raise the

crucible to a bright red heat; the steel is then introduced, the crucible covered, and the furnace filled up with coke. When the steel is melted, the crucible is lifted out with a pair of tongs, and its contents poured into a rectangular or octagonal mould of cast iron, which has been previously heated, and is placed vertically for the steel to be poured in. The mould is made in two halves, closely fitting together, so that it may be opened for the removal of the bar of cast steel, and is coated inside with coal-tar soot. The quality of the cast steel produced is in some measure dependent upon the temperature at which it is poured, so that an experienced workman is employed for the purpose.

If the steel be not kept long enough in the furnace it will *teem*, or pour 'fiery,' *i.e.* it will boil up in the moulds, owing to the escape of gas, and the ingot will be 'honey-combed' and useless; on the other hand, if it be left in too long it will teem 'dead,' and the bar produced from it will be of inferior quality.

At the present time large quantities of cast crucible steel are made from iron which has not been *converted*, *i.e.* has not had carbon imparted to it by the cementation, or, as it is termed in the trade, converting process. Malleable iron, cut into small pieces, is charged into crucibles, and melted with a certain amount of carbon, the proportion of which will depend on the *temper* required in the steel. Towards the close of the melting operation some *spiegel-eisen* (iron rich in carbon and manganese) is added. In other respects the process is similar to the melting of converted iron. In the production of mild steel, *i.e.* steel low in carbon, if plumbago pots be used carbon need not form part of the charge; a little *spiegel-eisen* is added towards the end.

Much scrap, consisting of crop ends, &c. of mild steel, produced by the Bessemer and Siemens processes, is largely used in the production of crucible cast steel, the scrap being simply melted down with the necessary quantity of charcoal and *spiegel-eisen*.

These methods, now largely used in the production of crucible steel, are to all intents and purposes the same as that employed by the Hindoos in the production of wootz, with the exception that they could not produce so high a temperature as is easily attainable in this country in an ordinary furnace. The addition of manganese is another point of difference in detail of some importance. This was first suggested and patented by Josiah Heath in 1839. He had been in the Civil Service in India, where he observed the influence which manganese had on the wootz steel. By the addition of manganese during the melting operation, in the form of an alloy with iron and carbon, or as oxide of manganese with carbon, sound weldable steel was produced from iron which could not previously be employed for the purpose. Before the introduction of manganese into the crucible steel manufacture only the best Swedish brands of iron were used.

The manganese acts beneficially in several ways, mainly by its having a greater affinity than iron for oxygen and sulphur. The action of manganese in improving the quality of steel is possibly analogous to that which may be supposed to be played by oxide of copper in the toughening process in copper smelting ; *i.e.* it combines with the impurities (sulphur and oxygen), and although they still remain in the mass of the metal, yet they probably exist there only in the shape of mechanically admixed oxide and sulphide of manganese, instead of being in chemical union with the iron itself.

The greatest enemy of good steel is phosphorus. If this element be present in even so small a quantity as 0.03 per cent., instruments having a good cutting edge cannot be produced from the steel containing it.

Numerous attempts have been made to produce malleable iron direct from the ore (without the intervention of the blast-furnace) in a more or less spongy state, and to convert it into steel by melting it in crucibles, and imparting the

requisite amount of carbon by the addition of pig, &c. The great relative bulk of the substance to be converted into steel, the ease with which it oxidises, and the difficulty of obtaining it in a sufficiently pure state, are all against the chance of success of this method of procedure. The expense becomes too great to pay commercially. The only process which shows any likelihood of overcoming these difficulties is that of Dr. C. W. Siemens, hereafter to be referred to (p. 190), which is still on its trial (1882).

Since the introduction of the Siemens regenerative system,* furnaces built on that principle have been largely used in the manufacture of crucible steel. The melting-chamber is a long trench with overhanging sides, arched horizontally and vertically, to prevent them sinking in, should the pillars between the ports give way. It is divided into several compartments, each holding six pots, arranged in two rows. Each of these chambers communicates by flues with the regenerators, which are situated on each side of the melting-chamber. The gas and air meet at a point about two feet from the entrance to the melting-chamber, into which they flow together, the gas being undermost, very little mixing of the gas and air taking place up to this point, so that the full heat of the combustion of the gas is developed around the crucibles.

Over each pair of pots is an opening in the arch, fitted with a fire-brick cover, which can be raised and lowered by a lever.

The economy of fuel by the use of these furnaces is said to be very great as compared with the old system, about a quarter of a hundredweight of small coal doing work for which previously three tons of coke were required. The pots stand very much longer, owing to the absence of dust, which in the old melting-holes rapidly corroded them.

Since in the *cementation process* the carburisation of the metal proceeds gradually from the exterior to the centre of

* See chap. on 'Fuel,' p. 64.

the iron, it is obvious that, by stopping the process after a certain time, the piece of metal will consist of wrought iron coated to a greater or less thickness with steel, a combination extremely useful for purposes in which a hard surface in conjunction with the power to resist sudden shocks is required, as, for instance, in the case of the mandrils of lathes. Cementation applied in this way is known as *case-hardening*. The articles to be case-hardened are placed in a fairly air-tight box, which is then filled up with some substance containing animal carbon, such as leather, horns and hoofs, bone dust, &c., which has been sufficiently charred to make it powder up. The box and its contents are then very gradually raised to a red heat, at which they are maintained until the required thickness of *casing* has been obtained. About half an hour after the temperature has reached the proper point, the case-hardening will have proceeded to a depth of a sixpence, and so on. It is sufficient for most purposes to convert to the depth of $\frac{1}{16}$ th of an inch. The presence of alkaline substances, such as soda or lime, assists the process. When a hard surface is wanted, which has not to stand much friction, it may be produced by simply heating the iron to redness, sprinkling over the surface with prussiate of potash, and then plunging the metal into cold water. In this way a very hard surface, but of hardly appreciable thickness, is obtained.

The principle of cementation may equally well be applied to the softening of carburised metal as to the hardening of iron. It is only necessary to reverse the conditions. The metal which it is desired to partially or wholly decarburise is heated in oxide of iron (iron scale or red hæmatite), when the oxide of iron or the air contained between its particles acts upon the carbon at the surface of the metal, producing carbonic acid, which diffuses into the metal, where, coming in contact with a further quantity of carbon, it is reduced to carbonic oxide, some of which gradually diffuses out again, to be again converted into carbonic acid at the expense of

the oxide of iron and the air, and so on. It is not essential that a substance containing oxygen and capable of being reduced by carbonic oxide should be employed; an inert substance, such as sand, will do perfectly well. The really essential point in the process is that the quantity of air surrounding the metal should be very small; only slightly in excess of that required to reconvert the carbonic oxide into carbonic acid. Were a greater quantity of air present, the removal of the carbon would be attended by a partial oxidation of the iron, destroying its toughness. The same thing results if the process be carried too far. Silicon, sulphur, and phosphorus do not appear to be seriously affected during cementation. The conditions are unfavourable to their removal, even supposing that, owing to the process being carried too far, they became oxidised.

At first sight it may appear difficult to reconcile the theories just put forward to account for the two opposite kinds of cementation. They are, however, perfectly in accord with the laws of chemical affinity and of dissociation. It has been supposed that in the carburising process carbonic oxide undergoes dissociation in the pores of the iron; it might not unnaturally be asked: Why does not the carbonic oxide produced by the reduction of the carbonic acid in the decarburising process also undergo dissociation, depositing carbon in the pores of the metal? We will endeavour to explain this apparent anomaly. At any given temperature and pressure an atmosphere of carbonic acid, placed under conditions favourable to its reduction to carbonic oxide, will undergo a certain definite amount of change, the extent of the change being entirely dependent on the temperature and pressure.

The iron in the two cases under consideration has, probably, nothing to do with the matter, except in the sense that it furnishes the spaces in which the gases are placed, under the necessary conditions of temperature and pressure to initiate their dissociation. If we assume as a working

hypothesis that the conditions in the heated iron are such as to enable dissociation of both carbonic acid and carbonic oxide to take place, then all that happens in each case is easily explained. It will be shown that this hypothesis and the facts of the case are in perfect agreement, and it may therefore be fairly considered to constitute a true theory of the two processes under consideration. It must be borne in mind that in the dissociation of a gas a portion only of it is dissociated, the percentage being regulated entirely by the temperature and the pressure, being independent of the actual amount of any other gas also present.

In carburising cementation the metal is surrounded with carbonaceous matter, and the conditions are therefore favourable to the maintenance of an atmosphere saturated in respect to carbonic oxide, but containing only a relatively small amount of carbonic acid, since any carbonic acid entering the region around the metal would be decomposed by the charcoal into carbonic oxide ($\text{CO}_2 + \text{C} = 2\text{CO}$), so long as the atmosphere remained unsaturated with the latter. At the commencement of the operation the interstices of the iron are charged with an atmosphere the nature of which will depend on the previous treatment of the metal during its manufacture. Unless this atmosphere not only coincide exactly in composition with that on the exterior, but also be at such a tension as to ensure dissociation not setting in, *diffusion* will commence in both directions. It may, then, be safely inferred that diffusion would take place. The atmosphere which would diffuse in would consist in great part of carbonic oxide, not only for the reasons already given, but also because the *diffusion equivalent* of carbonic oxide is greater than that of carbonic acid in the ratio of 1 : 0·8; so that supposing, for the sake of argument, the external atmosphere were saturated in respect both to carbonic oxide and carbonic acid, this equilibrium would be destroyed on their diffusing into the metal. The force of diffusion is inversely as the square roots of the densities of gases. Deville has

shown that at the ordinary atmospheric pressure carbonic oxide is sensibly dissociated at about a bright-red heat. Now, the temperature is the same within and without the metal, and unless the pressure be too great in the interstices of the iron, the carbonic oxide will undergo dissociation therein also. Our hypothesis is, that the conditions in the iron are favourable to the dissociation of the carbonic oxide, and that the carburisation of iron in the cementation process is effected by this means. The next point to be considered, in order to prove or disprove this view, is what becomes of the oxygen liberated on the splitting up of the carbonic oxide. The oxygen cannot combine with the iron in the presence of an atmosphere supersaturated with carbonic oxide; but it can combine with carbonic oxide to form carbonic acid, since this latter gas will be stable, *i.e.* cannot undergo dissociation, the quantity present being insufficient to form a saturated atmosphere. There will thus be a continuous dissociation of carbonic oxide, accompanied by a proportional formation of carbonic acid, which, however, can never form a saturated atmosphere, since it will gradually diffuse out from the metal, and, coming in contact with charcoal, be reduced to carbonic oxide, which, in its turn, will act as a carrier of a further quantity of carbon to the metal. The only limit to the reaction is that determined by the actual fusion of the metal at the temperature of the operation, owing to its fusing-point being lowered by its union with the carbon deposited in it.

Let us now turn our attention to what takes place in *decarburising* cementation. As already pointed out, the decarburisation is effected by the agency of carbonic acid. Dissociation in this case, in all probability, has nothing to do with the matter. For, in the first place, the quantity of carbonic acid existing in the atmosphere outside the metal must be very small, quite insufficient to constitute a saturated atmosphere; and, secondly, the atmosphere in the interior could never become supersaturated with carbonic oxide

since its formation takes place *in situ*, and would of necessity stop when equilibrium of tension is reached. As in the case of carburising cementation, this state of equilibrium is never reached, owing to diffusion taking place continuously. There is, therefore, nothing to interfere with the complete removal of the carbon from the metal. Carbonic acid diffuses in and is reduced by the carbon, and the carbonic oxide thus formed diffuses out, and, in contact with oxide of iron or air, becomes converted into carbonic acid, and so on, the carbonic acid acting as a carrier of oxygen. The assumption that the iron does not take a primary part in the reaction in the carburising process is supported by its evident inactivity in the decarburising process.

In speaking of the production of malleable cast iron, it was pointed out that, if the air were allowed to have too free access to the surface of the iron, oxidation of the metal would take place. This fact has recently been turned to practical account, it having been found possible to so oxidise the surface of the iron as to form a protective coating against the action of damp air, in other words, to prevent rusting.

Professor Barff, about ten years ago, patented a process for the protection of iron surfaces by forming on them a film of magnetic oxide. This is effected by heating the iron to redness, and passing superheated steam over it. The iron being more electro-positive than the hydrogen, decomposes the aqueous vapour, setting free the hydrogen, and taking its place in combination with the oxygen. Prompted by these results, Mr. Bower experimented to see whether the same result could not be arrived at by means of air, and he found that, by allowing air in suitable proportion to come in contact with iron heated to redness, a firmly adhering coating of magnetic oxide could be formed. If too much air is admitted a 'lobster' results, *i.e.* the article becomes coated with red peroxide of iron.

The process is carried out in the following manner: The articles to be coated are placed in a fire-brick chamber of

suitable dimensions, connected with a set of gas-producers by flues, in which the gas is allowed to mix with highly heated air, slightly in excess of that required for its complete combustion. The carbonic acid, which is at a very high temperature owing to the heat developed during its formation, and the small amount of free oxygen pass into the working-chamber, where they heat the articles up to the requisite temperature to cause them to combine with the oxygen. From the working-chamber the spent gas passes through regenerators used to heat the air required for the combustion of the producer gas. The amount of air admitted to the working is thus, it will be observed, perfectly under control. Over the exterior surface of the coating of magnetic oxide a film of sesqui-oxide forms. The foregoing part of the operation requires about half an hour. In order to remove the film of ferric oxide, carbonic oxide is next passed over the metal for about a quarter of an hour. The thickness of the coating will depend on the number of times these two operations are repeated. It is stated that for indoor work from three to four hours are sufficient, whilst for outdoor work an hour or two longer is required.

So far we have traced how certain methods for the manufacture of steel naturally grew out of the primitive process of extracting iron direct from its ore. It should now be clear that the difference between iron and steel is only one of degree, depending upon the amount of carbon present. It may safely be stated that iron has never been manufactured on anything like a large scale, free from carbon ; it always contains at least a trace under such circumstances. Until recently the term steel was only applied to carburised iron which had a high tensile strength, was elastic, and *would harden on being heated to a certain temperature and suddenly cooled*. This definition no longer holds good, for a vast quantity of slightly carburised iron produced by modern methods now goes by the name of steel. How this came about will be made clear in the sequel.

It is necessary that we should now return to the study of the processes by which iron can be extracted from its ore. The more important steps in early blast-furnace development were made in Carinthia and Styria. There the dimensions of blast-furnaces were gradually increased until the limit beyond which malleable iron could no longer be made was reached. Before this limit was arrived at, furnaces were in use in which either malleable iron or cast iron could be made by slightly altering the method of working. The same furnace was known as a *Stückofen* or a *Blauofen*, according to whether it was being worked for the production of malleable iron or cast iron. The height of these furnaces was from about 10 feet to 16 feet; in shape they commonly resembled two truncated cones placed base to base; sometimes, however, they increased in diameter regularly from top to bottom. Taking as an example a recorded case, we find that a furnace 16 feet high measured $1\frac{1}{2}$ feet at the top and $2\frac{1}{2}$ feet at the bottom, the diameter at the widest part, or 'boshes,' represented by the junction of the bases of the two cones, being 4 feet. This furnace was worked with one twyer, made sometimes of copper and sometimes of clay, placed about 14 inches above the bottom. When malleable iron was being produced, the lump was withdrawn from the bottom of the furnace through an aperture 2 feet wide, which was kept closed during the smelting, with the exception of a small hole through which the slag escaped. Much damage was often done to the furnace in withdrawing the lump of metal, owing to its adhering to the material forming the bottom, and tearing it away. From the description of the malleable iron obtained, it was evidently of a decidedly steely nature.

When it was desired to produce cast iron, the direction of the twyer was altered, and also the composition of the *burden*, the relative proportion of charcoal being increased. When it was intended to make malleable iron, the blast was made to impinge as much as possible upon the lump of metal, to

assist which the slag was tapped out often. These latter conditions would, therefore, be reversed in order to produce cast iron, and the metal would be exposed to the action of the charcoal for a longer time and at as high a temperature as could be obtained.

Even in some of the higher Indian furnaces cast iron was sometimes unintentionally produced in more or less quantity, much to the disgust of the workers, who looked upon this product as worthless. They were quite aware that the cause of cast iron forming is due to the temperature having risen too high. In making malleable iron in the blast-furnace, ores containing a considerable quantity of phosphorus might be employed, and yet the iron produced would be free from that objectionable element. If, instead of malleable iron, cast iron were made in the same furnace, all the phosphorus would pass into the metal.

The reason of this is that carbon has a greater affinity than phosphorus for oxygen. When smelting for cast iron, the charge consists largely of carbonaceous matter, therefore the phosphoric acid (P_2O_5) in the ore is soon deprived of its oxygen, and the phosphorus combines with the iron. Since carbon has a greater affinity for oxygen than phosphorus has, it also follows that, so long as the metal retains an appreciable quantity of carbon, the phosphorus cannot be removed to any great extent by oxidation.

When the blast-furnace was employed for making malleable iron, the conditions were favourable to the removal of phosphorus from the iron, the metal produced being sufficiently free from carbon to allow that portion of the phosphorus which had been reduced to reoxidise, and pass into the slag.

In consequence of the comparative ease and cheapness with which cast iron could be produced from the ore as compared with malleable iron, some simple means were naturally sought by which it could be converted into malleable iron. This was soon accomplished, in spite of the

presence of phosphorus, and gradually cast iron became the sole product of the blast-furnace.

In the furnaces we have so far considered charcoal was the only fuel employed. In course of time charcoal began to get scarce, and people were forced to consider whether coal could not in some way be used as a substitute. At first it was used only for forging iron, but in time, after many failures, it was successfully applied to smelting in the blast-furnace. The first to do this was Dud Dudley, about the middle of the seventeenth century. What means he employed are not known; he was greatly persecuted through the jealousy of other ironmasters, and he ended his days in poverty, his secret dying with him. There is not much doubt that his secret consisted in converting the coal into coke, and employing a good blast in his furnace. Although not absolutely lost sight of, the question of substituting coal for charcoal was not further practically developed until about a hundred years after Dudley's death. About the year 1730, one Abraham Darby took the matter in hand, and achieved success by treating the coal in the same way as in making charcoal from wood, *i.e.* he converted the coal into coke.

With the introduction of coke commenced a new era in blast-furnace history. The bellows which had up to that time been employed were not equal to dealing with the more refractory fuel. This resulted in the introduction of blowing-cylinders worked by water power, and in due course of time by steam power.

At first single-acting cylinders were used, but in time double-acting cylinders were substituted for them. The arrangement is a very simple one. The cylinder, which may be of cast iron, is fitted with an air-tight piston, to which a reciprocating motion is imparted, causing air to be drawn in and forced out alternately, through suitably arranged valves in each end of the cylinder. The air forced through the outlet valves of the blower passes into pipes attached to both ends of the cylinder, and connected

with the blast main which supplies the furnaces. The arrangement of the valves and the attitude of the blowing-cylinder to the motive power have been varied in many ways ; the principle involved must always remain the same. The blowing-cylinders attached to blast-furnaces of the present day are often required to discharge 50,000 cubic feet of air per minute, and sometimes even more, at a pressure varying from about $3\frac{1}{2}$ lbs. to 10 lbs. per square inch, and even higher under special circumstances. In order to steady the blast a large chamber, equal in capacity to from twenty to fifty times the volume of blast passed into the furnace per second, was interposed between the blower and the furnace. The blast mains are now commonly so large that generally no other regulator is required. Valves are placed in the regulator or blast main to prevent the pressure of the blast rising beyond a certain fixed amount.

Although the discovery that coke could be used for the manufacture of iron was a very important one in consequence of the increasing scarcity of charcoal, yet it must not be lost sight of that, owing to coke containing always more or less sulphur, it is really not so well suited as charcoal for the production of the best malleable iron. Even at the present day, where it can be obtained, charcoal is still used, notably in Sweden, and parts of America. The use of charcoal must, however, sooner or later come to an end.

The most important innovation which has ever taken place in blast-furnace practice is undoubtedly the introduction of *hot blast*. This took place in the year 1828. It was first carried into effect by Neilson, at the Clyde Iron Works, and notwithstanding that the idea was received with much derision at first, yet by about the year 1835 it had been universally adopted throughout Scotland. As a direct consequence of the use of a heated blast, it became possible to employ coal in the furnace, the heat being sufficient to cause the coal to coke to the necessary extent whilst still near the top. All coal cannot be used with equal advantage :

a highly caking coal would be objectionable. The splint coal of Scotland is well suited for use in blast-furnaces; in the Staffordshire district coal is also used; in Wales and America anthracite is similarly employed, generally, however, in admixture with other coal; in Yorkshire, Durham, Cumberland, and some other localities, the blast-furnaces are fed with the best oven-coke procurable.

Irrespective of the nature of the fuel used, the effect of heating the blast was to cause considerable saving in fuel, and also to increase the actual *make* of a furnace; the latter being to a great extent the direct consequence of the former, for to decrease the proportion of fuel in the burden is equivalent to increasing proportionally the ore.

By blowing in hot air the temperature in the vicinity of the twyers is sensibly increased. With cold blast a *nose* of chilled slag forms at the end of the twyer, and extends itself to a greater or less distance into the furnace according to the temperature, thereby modifying the working and affecting the quality of the metal produced. With hot blast it is necessary to cool the twyers by passing a stream of water through a jacket surrounding them, to prevent their being fused. In the case of hot blast it is clear the conditions are more uniform and more under control; atmospheric influences can less affect the working of the furnace, and the zone of fusion is brought lower down in the furnace and concentrated within a smaller area. The importance of these considerations is apparent when we call to mind the conditions in the blast-furnace. A tall cylindrical chamber is filled with ore, fuel, and flux, in certain proportions; at the base air is blown in, which, coming in contact with incandescent fuel, combines with it, forming carbonic acid so long as the air is in excess, and carbonic oxide in the higher regions, where the influence of the fuel predominates. Part of the carbonic oxide as it passes through the ore reduces it, with formation of carbonic acid, in the way already explained; another part undergoes dissociation in

contact with metallic iron, in the interstices of the partially reduced ore. The carbonic acid thus formed is reduced again to carbonic oxide on coming in contact with more fuel, as it passes up; so that a supply of carbonic oxide is maintained throughout the furnace. It has been shown that, in the primitive blast-furnaces, the earthy matter of the ore was fluxed off by means of oxide of iron. In large furnaces producing cast iron this would be impracticable, for oxide of iron forms an easily fusible 'scouring' slag, which runs down into the *crucible* of the furnace, and meeting with the cast iron which is collecting there, decarburises it more or less, so that an inferior white cast iron is produced, instead of a grey iron rich in carbon. Besides, oxide of iron could not remain unreduced unless an insufficiency of fuel were employed. In modern blast-furnaces these difficulties are avoided by using lime (CaO) as a fluxing agent. It forms a slag requiring a much higher temperature to render it fluid than is the case with oxide of iron, and, owing to the great affinity which calcium has for oxygen, it is not reduced, except in very minute quantity, either by the carbonic oxide or by the carburised metal. In the higher zones of the furnace there exists, then, the following state of things: the ore has become reduced, and carbon, in a fine state of division, has been deposited throughout the mass; at the same time, the earthy matter has combined with the lime to form a slag of pasty consistency; besides the reduced ore and slag there is also a considerable quantity of unconsumed fuel. All that remains to be done, then, is to raise the temperature of the charge, so that the iron and slag may be rendered liquid, and thus enabled to separate one from the other by reason of the difference in their specific gravities. This is effected by burning the intermixed carbon by means of a powerful blast of air. It requires but a moment's reflection to see that, as the necessary temperature for fusion is obtained by means of air introduced into the furnace, there would be a

great disadvantage in the fusion being effected *above* the point at which the air is introduced, for in that case the already fused metal would have to pass in front of the twyers, and in doing so would become to some extent decarburised. There is another point to be borne in mind, viz. that the carburisation of the metal is not effected alone by the carbon deposited in it. It will be recollected that when iron and carbon are brought together at a high temperature, the former dissolves the latter. That the amount of carbon ultimately taken up by the metal is to some extent dependent on this reaction is probable. It follows that, within certain limits, the higher the temperature brought to bear upon the mixture, the more highly charged with carbon will the metal become. It is not uncommonly stated that there is no limit to the advantage to be obtained by raising the temperature. In this, as in everything, there is a well defined limit, though our ignorance may obscure it. In the first place, there is a limit to the amount of carbon which the iron is capable of taking up. Once that the slag and the metal are in a sufficiently liquid state, and the latter has taken up the maximum amount of carbon in the graphitic state, what further advantage can there be in raising the temperature? Surely, the only result would be to extend the zone of fusion higher up in the furnace, which would be disadvantageous, and the wear of the lining of the furnace would be increased.

Until within the last few years, the blast was heated by passing it through iron pipes heated from the exterior ; recently, however, fire-brick stoves, heated by the waste gases of the blast-furnace, have come largely into use for the purpose. (See chapter on 'Fuel,' p. 70.)

At first, and for some years, the blast was only heated to a few hundred degrees, but since, the temperature has been gradually increased, until at present it is commonly employed at 900° F., and sometimes at 1,100° or 1,200°. The Cowper stoves are said to be capable of raising the temperature

of the blast to $1,500^{\circ}$, but they are not generally worked at their maximum power.

The utilisation of waste gases—a source of great economy—has only been carried into effect within the last half-century. At first it was only sought to utilise the sensible heat, a very small quantity as compared with that to be obtained by the combustion of the carbonic oxide and hydrogen contained in the gases passing out at the top of a blast-furnace, amounting in the case of carbonic oxide to 25 per cent. by volume on an average, the hydrogen varying up to 7 or 8 per cent.

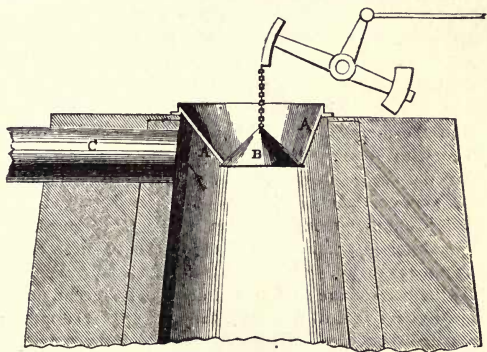


FIG. 21.—Cup and Cone for closing the Blast-furnace, in order that the waste gases may pass into the lateral flue, as shown by the arrow.

Various means have been tried with the view to collect the waste gases without interfering with the necessary arrangements for properly charging the furnace. That universally employed at the present day is the cup and cone (Fig. 21), or some modification of it. The ore, fuel, and flux are tipped from barrows into the annular space or cup (A), and the cone (B), which is attached by means of a chain to a lever, is lowered, allowing the charge to fall into the furnace, after which it is raised again, so as to entirely close the top

of the furnace. The gases are drawn off through the outlet (c). This arrangement has several great advantages. In the first place, it is very simple, and it can readily be altered without interfering with the body of the furnace ; it distributes uniformly the burden in the furnace ; and, by altering the dimensions of the cone, the arrangement of the charge can be regulated to suit the working of each furnace. As the charge descends more rapidly in the centre than at the sides, it is essential to the proper working of the furnace that the larger material should be in the centre, so that the current of gases may ascend freely in that direction, and not creep irregularly up the sides. This the cup and cone accomplishes more satisfactorily than any other system which could be adopted. In one modification the exit tube for the gases is attached to the apex of the cone.

With the increase in height of blast-furnaces it became necessary to have resort to mechanical contrivances for raising the charge to the top. For this purpose inclined planes were commonly in favour. They consist in a staging reaching from the ground to the top of the furnace, supported at several points, and inclined to the horizontal at an angle of about 25° to 30° . On this incline travels a carriage, on which the barrows are sent up and down. In recently-constructed works vertical lifts are usually put up. The motive power may be either steam, compressed air, or some hydraulic arrangement. Of these, the first is that most commonly employed. Perhaps the simplest method is that in which two drums are fixed on the same shaft, which is revolved by means of a small pair of reversing engines. Wire ropes attached to each drum pass over a pulley at the top of the furnace, and are connected with cages working between vertical guides. As one cage goes up the other is coming down, so that, to some extent, they balance one another. Amongst the pneumatic arrangements that of Gijers is the most in favour in the Cleveland district. In a cylinder, about 36 inches in diameter, works a cast-iron piston

or ram, attached to which are wire ropes, carried over pulleys, and connected with the cage at its four corners. The weight of the piston is made to exceed that of the empty lift, but

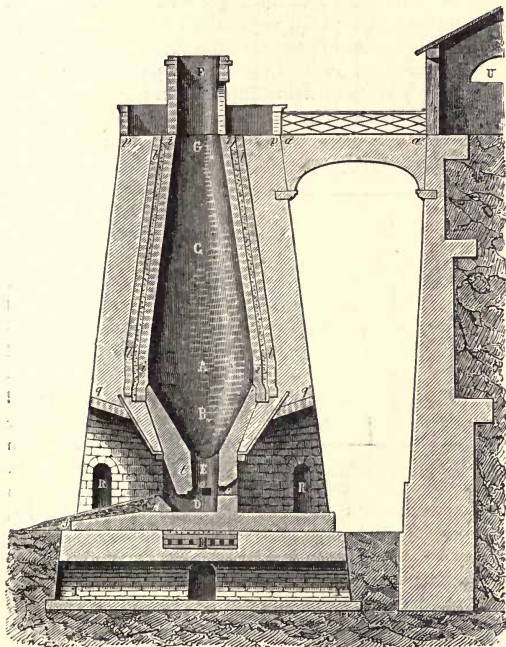


FIG. 22.—Blast-furnace for Smelting Iron Ores.

only by such an amount that when the lift is loaded the latter shall then be somewhat heavier. Thus, when the platform has been unloaded, it is only necessary to apply a few

pounds pressure to the piston in an upward direction to cause it to rise, and bring down the platform or *cage*; similarly, in order to raise the loaded platform, the piston is lowered by slightly exhausting the air from beneath it. The pressure and partial vacuum are readily obtained, as required, by means of pumps worked by a small engine.

Blast-furnaces have been varied in shape and dimensions considerably in different localities, and in the same locality at different times. Fig. 22 will give a general idea of the way in which they were constructed up to quite recent times. Furnaces of this type of construction were to be seen in all

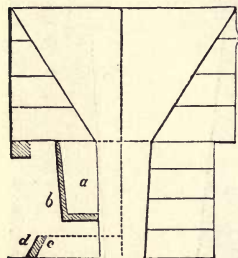


FIG. 23.—Boshes, Hearth, and Crucible of Blast-furnace.

parts of Europe. The body of the furnace is lined with some refractory sandstone, more generally with fire-clay (*i*); between this and another similar layer (*l*) is a layer of powdered coke or sand, which helps to prevent the furnace being pulled to pieces by the unequal expansion and contraction. It will be observed that the furnace gradually widens for a certain distance, and then contracts.

From the widest part (A) down to the crucible (E) is termed the 'boshes.' The tendency in more recent constructions has been to make the angle of the boshes less acute. E is the *hearth* or *crucible*, and *o o* points at which the twyers, varying generally in number from 3 to 5, are introduced. Three sides of the hearth are occupied by the twyers; the fourth is arranged for the tapping-out of the metal and slag.* This is shown in Fig. 23, *a* being a heavy block of stone, called the *tym-p-stone*, supported by a cast-iron *tym-p-plate* (*b*), built into the masonry of the furnace, whilst the lower part is enclosed by the *dam-stone* (*c*), faced

* From the Scandinavian *slag*, *dross*.

externally with a thick cast-iron *dam-plate* (*d*). Both the *tymp-plate* and the *dam-plate* are kept cool by a current of water circulated through them. The space *c* is termed the *fore-hearth*. The metal is tapped out through a hole at the bottom of the plate *d*, the cinder being allowed to flow through a notch at the top of the plate and down an incline into the cinder-tubs, where it solidifies, and is removed to the tip-heaps. The tap-hole for the metal is often placed to one side of the *dam-plate*, so that it can be easily raised, if the furnace should be working badly. It was customary, after each tapping, to partially fill the *fore-hearth* with ashes, which floated on the metal and protected it; when the metal was tapped out the ashes and slag settled down to the bottom of the *fore-hearth*, necessitating much labour and loss of time in removing it each time. This difficulty was got over by covering in the top of the *fore-hearth* with clay, weighted down to keep it in place, a hole being made in it for the exit of the slag. It is very common now to do away with the *fore-hearth* altogether, the slag being allowed to flow out from the crucible, at a point 15 to 18 inches below the *twyers* through a water-cooled block, another hole being provided at the bottom of the crucible for tapping the metal out. When it is desired to close the tap-hole, a lump of clay, held on the end of an iron bar, is forced into it. It is very important that the clay should not be too easily fusible, nor, on the other hand, be too refractory. In the former case it would be liable to fuse out, whilst in the latter it would give rise to a 'hard tap'—*i.e.* great difficulty would be experienced in making a hole through it with the iron bar in order to tap. A mixture of 2 parts of fire-clay and 1 of small coal, ground together, is said to answer well.

Fig. 24 illustrates a modern type of furnace, as constructed at Messrs. Bolckow, Vaughan & Co.'s in the Cleveland district. It will be observed that light iron columns are substituted for the massive brick structure in the older furnace, thus giving more working space, and

enabling the twyers to be readily got at. A strong circular

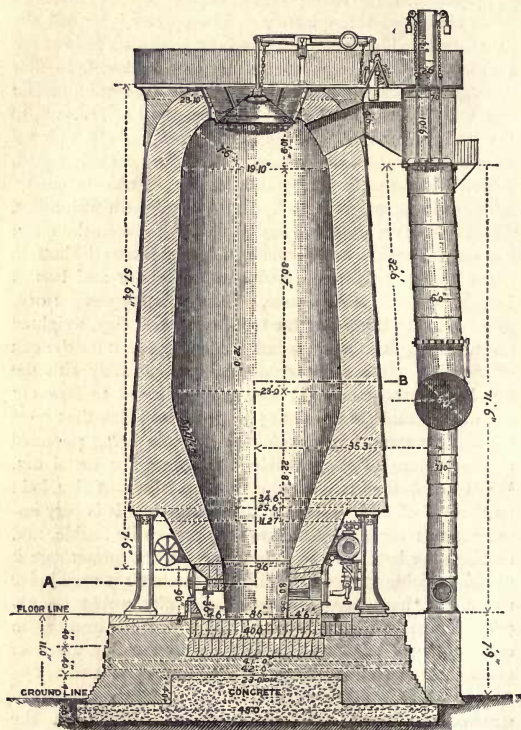


FIG. 24.

iron casting, made in segments, rests on the columns, and supports the body of the furnace, which is constructed of

the best fire-bricks, and encased in wrought-iron plates, or braced with hoops. The waste gases pass from the top of the furnace, through the 'gas box,' into the vertical tube, called a 'down-comer,' and thence are drawn off into the main (shown in section at B), which may be connected with a number of furnaces, and from which the gases are led

SECTION ON LINE. A. B.

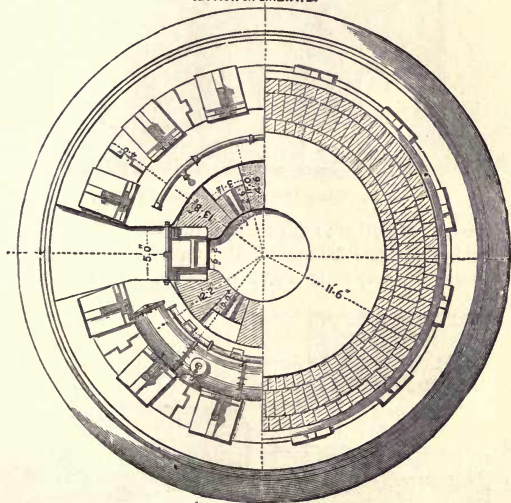


FIG. 25.

into hot-blast stoves, and under boilers, where they are mixed with the requisite quantity of air, and burnt for heating the blast and raising steam, &c. At c (Fig. 26) is shown one of the *twyer houses*, in which rest the twyers through which the blast is conveyed into the furnace by means of the blast mains (B), connected with the 'horse-shoe' main (A).

At the angle of the blast-main pipes is inserted a piece of mica, through which the temperature of the furnace can be observed. The twyers generally consist of a wrought-iron coil (Fig. 27), encased either in clay or cast iron (Fig. 28).

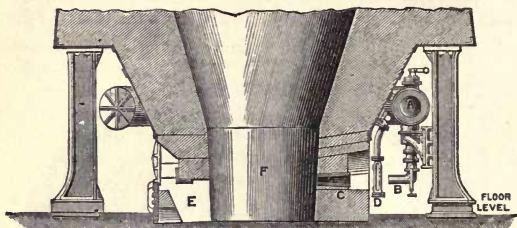


FIG. 26.—Enlarged Section of lower part of Furnace.

The only objection to this form of twyer is that, if the nose accidentally gets burnt off, the water, which is flowing through the coil under considerable pressure, is suddenly



FIG. 27.

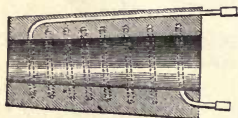


FIG. 28.

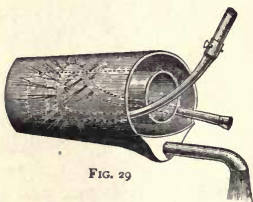


FIG. 29

forced into the furnace, causing an explosion, which sometimes greatly deranges the working of the furnace, and occasionally does considerable damage. In Lloyd's spray twyer (Fig. 29) this defect is got over. The only objection

which has been raised to this system is, that the holes in the spray-tube are liable to become blocked up. This difficulty is easily overcome by filtering the water first through a copper tube with one-eighth inch holes, which can be cleaned out, when necessary, by means of a simple sludge-cock arrangement. The size of the twyers varies very considerably ; they must be proportioned to the volume of the blast, so that it may be forced well into the centre of the furnace, and not creep up the sides, which would cause irregular working. The average pressure of the blast is about $4\frac{1}{2}$ to 5 lbs. at the twyer, in most districts in this country. When anthracite is used a higher pressure is required ; with charcoal the pressure must be lower.

A steady improvement has been going on in the construction of blast-furnaces, since the time when they first began to make cast iron up to the present day. Instead of 10 feet high, blast-furnaces may now be seen 100 feet high, but it has been shown that these furnaces have passed the limit of economy to be obtained in that direction. The height of a blast-furnace must be regulated by the natures of the ore and fuel to be used in it. Experience has shown that in no case is anything gained by increasing the height beyond about 80 feet. The furnace illustrated in Fig. 24 is 72 feet high, and the diameter of the boshes is 23 feet. The usual height, in England, for furnaces, working on Cleveland, Cumberland, Scotch, or Spanish ore, is from 70 to 75 feet, with 18 to 20 feet diameter at the boshes ; good coke, splint coal, or anthracite being used. If the ore or the fuel tend to crumble under pressure, the height of the furnace must be proportionately less. There appears, however, to be no necessity to reduce it below 50 feet when coke or coal are used.

In some cases the ore is calcined, and charged into the furnace whilst still hot. This is never done with the rich hæmatites of Spain and Cumberland, but only with those

ores which contain carbonic acid in combination with the iron, such as, for example, the Cleveland stone and the Scotch ores. The effect of calcination is to drive off the carbonic acid and moisture, thereby rendering the ore more porous and easily acted upon by the reducing gas in the blast-furnace. Calcination formerly was always carried on in heaps. This method gives very irregular results, owing to the variable action of atmospheric influences, some of the ore being hardly affected, whilst other parts have actually become fused. Kilns of some kind or other are now almost invariably employed, in which plan there is great economy. The trucks which have conveyed the ore from the mines or the docks are carried on a line of rails over the kilns or calciners, into which the ore is dropped, a certain amount of small coal being thrown in at the same time. In Fig. 30 is shown the calciner commonly used in the North of England. The calcined ore is drawn out, as required, at the bottom of the calciner into barrows, in which it is sent up, still hot, to be charged into the blast-furnace. Thus space and labour are economised, the ore is uniformly calcined, and but little heat is lost in transferring it to the furnace. Besides removing the carbonic acid and moisture, the effect of roasting is to convert the protoxide of iron into peroxide, thus lessening the tendency of the ore to fuse and become less easily permeable by the reducing gas. Protoxide of iron combines with silica at a comparatively low temperature whereas peroxide of iron does not.

There is practically very little advantage, and often none, in calcining the limestone, for although the carbonic acid is thereby driven off, more or less, according to the care taken, yet, at the low temperature which exists at the top of the furnace, the burnt lime is capable of combining with the carbonic acid with which it then comes in contact, so that it is brought back to a great extent, if not entirely, to its original condition.

The use of lime has an additional advantage, beyond that

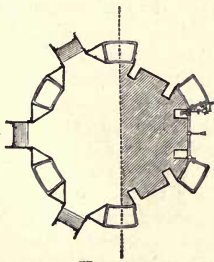
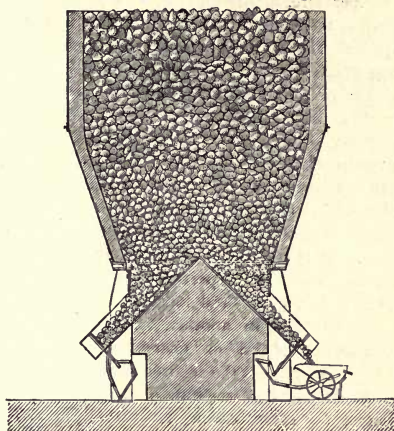


Fig. 30.

of being a highly suitable fluxing agent ; by its means a considerable proportion of the sulphur in the ore and fuel

is retained in the slag. The most favourable conditions are a high temperature, plenty of lime, and plenty of fuel. It would appear that, under these circumstances, the carbon, the sulphide of iron, and the lime (calcium oxide) react upon one another, the oxygen of the lime passing to the carbon, and the sulphur, leaving the iron, unites with the calcium. The higher the grade of the iron the less sulphur will it contain.

The relative proportions of ore, fuel, and flux in the burden of a furnace must depend upon the nature of the ore and flux, and upon the dimensions of the furnace, the temperature of the blast, and the grade of iron it is desired to produce. It is customary to keep the weight of the fuel constant, and vary the proportion of the ore as required. For this reason, the ore and limestone are placed in one barrow and the fuel in another. A certain number of barrowsful form a 'round' or 'charge.' Thus, for a furnace making Bessemer pig, the charge might consist of six barrows of hæmatite ore, weighing 9 cwts. each, with $1\frac{1}{2}$ cwt. of limestone on each barrow, and six barrows of coke, averaging $5\frac{1}{2}$ cwt. each. If much silica be present in the ore, there should be added to each barrowful, about $\frac{1}{4}$ cwt. of aluminous ore.

The slag from blast-furnaces consists essentially of silica, alumina, lime, and magnesia. Taking the general run of cases in England, the silica varies between 30 and 40 per cent., being generally above 35 per cent. rather than below. The same figures will apply to the percentage of lime. The alumina ranges between 12 and 22 per cent., and the magnesia averages 5 to 7 per cent. In some cases these figures are considerably departed from. There is generally more or less manganese present : this varies greatly. The alkalis, potash and soda, generally amount to 1 or 2 per cent. ; sulphur is always found in the slag, the amount depending on the quality of the ore and fuel, and on the working of the furnace.

The earthy matter, or *gangue*,* of iron ores usually consists mainly of clay, and it is therefore sufficient to add limestone alone to flux it. Occasionally, however, when a large proportion of silica is present, it is desirable to add alumina in some form or other, aluminous iron ore, such as that obtained from near Belfast, and other localities in Antrim, being commonly employed for the purpose. In order to form a satisfactory slag the lime should be present, at least, to the extent of one-third the weight of the constituents of the clay ; and since limestone (carbonate of lime, CaOCO_2) contains rather more than half its weight of lime (CaO), two parts of limestone should be employed for three parts of clay. It has already been explained that, in practice, in order to prevent sulphur from passing into the metal, more lime is used than actually required as flux ; it is, however, also essential for the removal of sulphur that the charge should contain a large proportion of fuel, and that the furnace should not be making white iron.

The quantity of slag produced in the extraction of each ton of pig iron will vary with the richness of the ore under treatment ; roughly speaking, it may be taken at from 10 to 30 cwt.

The colour of slag varies greatly according to the ore being used and the grade of iron being produced. A light burden, *i.e.* one in which there is a large proportion of fuel to ore and flux, is favourable to the production of a light-grey slag ; whilst a heavy burden would tend to produce a slag which, according to circumstances, might be yellow, green, brown, or black. To some extent the colour and consistency of the slag indicate the kind of iron being produced, but, to form a trustworthy opinion, it is necessary to be well acquainted with the general working conditions of the furnace.

The physical characters of the same slag vary greatly, according to the way in which it has been treated, just as is the case with glass. Quick cooling tends to produce a

* From the Northern word *Gang*, a mineral vein.

vitreous, more or less transparent slag, with a conchoidal fracture; slow cooling causes the same slag to be dull, opaque, and stony in appearance. Slags containing much lime or magnesia are the least fusible; they are stony-looking, whether quickly or slowly cooled, and soon disintegrate on exposure to damp air. The fluidity of slag is greatly increased by the presence of oxide of iron or of manganese, especially of the latter.

Various means have been suggested for utilising blast-furnace slag; so far none of them have been taken advantage of to any great extent. Very excellent bricks, cement, concrete, and mortar are now made from slags, especially from those rich in lime, such as the slags from furnaces producing Bessemer pig, *i.e.* metal for subsequent treatment in the Bessemer converter. It is usual to mix some fresh lime or lime-cream with the granulated slag, say one part of the former to four or six parts of the latter, and then mould the mixture in some form of brick-making machine. By grinding the calcareous slag very fine, it may be used successfully without any addition of fresh lime. Bricks made by these methods, after having been exposed to the air for a few weeks, become as hard as stone. Highly calcareous slag has been employed to some extent in the manufacture of glass, its freedom from oxide of iron making it well suited for that purpose. For steam-packing, 'slag wool,' obtained by blowing air through the molten slag as it flows from the furnace, has been employed to some extent. Slag wool also forms a useful filtering agent. There is one very important objection to the use of slag wool for many purposes for which it would otherwise be useful, *viz.* that the fine filaments are extremely brittle, and the material being very light, the needle-like particles are liable to become diffused in the air of a room, and pass into the lungs and eyes, where they would cause very serious mischief. Slag containing a large excess of lime falls to pieces on exposure to the atmosphere, and is therefore unsuited for

the production of 'wool.' Attempts have been made to utilise slag for manure, on account of the large quantity of lime and potash it contains ; its application to this purpose has not, however, been successful. It must be borne in mind that the lime is almost entirely in combination with silica ; one might as well use bottle-glass, and expect it to improve the land.

The great cost of transport of slag, and of articles made from it, is the principal cause of the failure of attempts to utilise this waste product. Ironmasters would be only too glad to let anybody have it for nothing, on their undertaking to remove it from the works. Many millions of tons of slag are produced in this country annually.

For the sake of convenience, the carburised iron produced in the blast-furnace has hitherto been spoken of as cast iron. In practice, this term is only applied to the product of the blast-furnace after it has been remelted, and *cast* in a foundry. Carburised iron which has not undergone this treatment is known as *pig-iron*, or in works as *pig*.

From what has been said so far, it will have been gathered that the iron produced in a modern blast-furnace contains more or less carbon; all the phosphorus originally in the ore, and a certain proportion of sulphur derived from the ore, and fuel. Besides the foregoing substances, pig-iron always contains more or less silicon, which has been reduced from silica at a high temperature, by carbon or carbonic oxide, and entered into combination with the iron. The higher the temperature employed, and the greater the proportion of fuel in the burden, the more silicon will there be in the pig. The taking up of silicon by pig-iron is also greatly favoured by the presence of a considerable quantity of alumina in the charge, as well as by the presence of alkaline silicates. Silicious pig-iron finds an important application in the production of steel castings of Bessemer or Siemens metal, free from honeycomb.

Other substances (e.g. magnesium, calcium, copper,

arsenic, cobalt, nickel, titanium, vanadium) may occur in pig-iron in small quantity; their influence is not, however, of such general importance as to warrant any special reference to them in this work.

Manganese is also a common constituent of pig-iron, in greater or less quantity according to the ore used and the conditions in the furnace. Manganese has been made to replace iron in pig-iron to the extent of 85 per cent. Pig-iron containing from about 5 to about 20 per cent. of manganese is known as *spiegel-eisen*; with a greater percentage of manganese it is called ferro-manganese. These two alloys are now largely manufactured for use in the Bessemer and the Siemens processes.

With small amounts of manganese, beginning with about 7 per cent., the alloy crystallises with large brilliant facets: hence the name *spiegel-eisen*; the crystalline structure, after a certain point, becomes smaller with each increase in the percentage of manganese, finally becoming very fine grained. Ferro-manganese, at first, was made by smelting together carbonate of manganese and a small quantity of pig-iron in a Siemens' furnace. Since the introduction of very rich mangiferous ore from the South of Spain, the conditions favourable to the production of ferro-manganese in blast-furnaces has been carefully studied, and all difficulties surmounted, so that now alloys containing 70 to 85 per cent. can easily be produced in this manner. There is no difficulty in reducing the whole of the manganese from an ore, but in practice it is found impossible, in that case, to work the furnace so as to produce a good *spiegel*, for, if all the manganese be reduced, the slag becomes 'dry' and infusible, unless a much larger quantity of fuel be used, in which case the metal would become grey, and useless for the purposes for which it is required. By adjusting the burden so as to have rather less lime than would form a basic silicate with the silica, and regulating the proportion of fuel, just sufficient oxide of manganese can be allowed to pass into the slag to

render it liquid. The slag should be green and vitreous. There is a decided advantage in working with a blast at a high temperature and pressure. A greater quantity of coke is required than in making ordinary pig, the reduction requiring a higher temperature. The 'make' of a furnace is also very considerably less when working on rich manganese ore than when making ordinary grades of pig-iron.

The magnetism of an alloy of iron and manganese is less the greater the percentage of the latter present.

Chromium pig-iron can be made in the blast-furnace by the use of chrome iron ore. With 10 per cent. the alloy is white and brittle, the fracture being close grained and irregular. *Tungsten* can also be alloyed with iron in the blast-furnace by adding the mineral wolfram—a tungstate of protoxide of iron and protoxide of manganese.

The carbon in pig-iron cast in the pig beds in the ordinary way may exist in two distinct conditions, viz. the uncombined, or *graphitic*, and the combined, the relative proportion of each depending on the amount of fuel in the burden, and on the temperature to which the metal has been raised. About $4\frac{1}{2}$ per cent. is the maximum amount of carbon which iron is capable of taking up in the blast-furnace in the absence of manganese. With the maximum amount of carbon the pig exhibits a highly crystalline fracture, and on being pulverised scales of graphite can readily be separated; the colour of a fractured surface is bluish-grey; the metal has very little tenacity, breaking easily with a dead sound. As the amount of carbon decreases the grain becomes finer, the colour lighter, and the tenacity increases; but the fluidity of the metal when molten decreases. With a still further decrease in the proportion of carbon the iron becomes mottled, the groundwork being greyish-white, with grey spots interspersed in greater or less number over its surface. Below this point the iron becomes white, the greater part of the carbon then being in the combined condition. The

presence of manganese promotes the taking up of carbon by iron, facilitating its remaining in combination, so that a pig-iron containing manganese may be white, although the carbon may amount to from $4\frac{1}{2}$ to $5\frac{1}{2}$ per cent. Nevertheless, it has been shown that iron containing as much as 16 per cent. of manganese, produced at a very high temperature, may be grey. The presence of sulphur very materially affects the colour and quality of iron. If sulphur be added to grey iron, it has the effect of throwing some of the carbon out from its state of combination. Thus, when about 3 per cent. of sulphur is melted with grey pig-iron, an extremely hard white metal is produced, the carbon separating in a sooty form from the molten metal, and floating up to the top. With less sulphur, say about $\frac{1}{2}$ per cent., a strong mottled iron is obtained, possessing great strength. The conditions which are favourable to the production of a low grade of iron are also those which favour the retention of sulphur by the metal ; on the other hand, the higher the grade the more silicon will there be present.

It has been observed, that in pig-iron which has been heated considerably above its fusing point the carbon tends to separate in the graphitic form, whereas the same metal heated to a lower degree would, on cooling, retain more carbon in the combined condition. The reason of this is not at present understood. The fact is an important one, showing the necessity for studying carefully the question of temperature in blast-furnaces. Two furnaces might be making, from the same ore, metal containing exactly the same total amount of carbon, yet the quality of the product in the two cases might be very different, owing to variation in the conditions of the carbon and iron, in consequence of the temperature not being the same in the two cases. From what has been said regarding the working of the blast-furnace, the difficulty there is in obtaining exactly the same grade of iron from a furnace for any length of time together will be understood.

Grey iron passes directly from the solid to the liquid state on being heated ; but white iron, not containing manganese or much phosphorus, becomes pasty before liquefying. The difference in the condition and amount of the carbon is the cause of this. The higher the grade of iron the more malleable will the metal be ; the most highly carburised grey pig may be considered to consist, practically, of a mixture of malleable iron and scales of graphite, for by crushing the pig, and levigating it with water, the graphite can be separated in a pure condition from the iron.

Since in grey cast iron a smaller proportion of the iron is in combination with carbon, and more of it in the true metallic state, this variety would be expected to exhibit more of the properties of metallic iron, whilst white iron ought to present the characters of the chemical compound of carbon with iron, described above. Accordingly, the grey cast iron is much softer than white iron, and admits of being filed and turned ; whereas the white iron is so extremely hard that a file will scarcely touch it, and a blow from a hammer, which indents a pig of grey iron, will break up one of white iron. The larger proportion of metallic iron contained in the grey cast iron causes it to require a higher degree of heat before it begins to exhibit signs of fusion, but it is capable of becoming very liquid at a sufficiently high temperature, so as to be easily run into moulds ; white cast iron, on the other hand, is softened at a rather lower temperature, but does not flow well, assuming a somewhat viscid or pasty consistence. It *scintillates*, or throws off sparks or *jumpers*, as it runs from the furnace, to a much greater extent than grey iron. White cast iron is about $\frac{1}{20}$ th heavier than the grey variety, its average specific gravity being 7.5, whilst that of grey iron is 7.1. The grey iron rusts more easily in air, and is more readily acted on by acids, than white iron, which may be ascribed partly to its containing more iron in an uncombined form, and partly to the acceleration of chemical action caused by the voltaic disturbance excited

by the contact of the particles of graphite with the particles of iron, in the presence of an acid (in the case of air, carbonic acid).

Manganiferous and phosphoriferous pig-irons become fluid more easily than ordinary white iron, and do not pass through the pasty stage.

The effect of phosphorus in pig-iron being to render it more fluid, this is taken advantage of in selecting metal suitable for working fine castings. It may be observed, that although the effect of phosphorus is to somewhat weaken the actual strength of cast iron, yet the ultimate strength of a casting, as a whole, may be increased by its presence, owing to the ease with which the metal takes the shape of the mould, and the greater soundness of the casting. The object which a founder has to keep in view is to use the cheapest metal consistent with obtaining in the casting the requisite properties for the purpose to which it is to be employed. This he does by keeping in stock a certain number of pig-irons of different but known grade, which his experience enables him to mix in the best proportions for whatever work he may have in hand.

The rate at which cast iron is solidified affects its properties to as great an extent as differences in its chemical composition. Thus, if grey pig-iron be melted, and then rapidly solidified by cooling it in contact with a metallic surface, it will, to a greater or less depth, according to its thickness and that of the metallic surface, be converted into white iron. This treatment is known as *chilling*, the metal produced by it being termed *chilled* cast iron, and the mould a *chill*. This modification of cast iron is extremely hard, and possesses great tenacity as compared with the grey iron from which it is produced, but it is very brittle. This latter property does not, however, in practice materially interfere with the applications for which the other qualities of this metal render it particularly well suited, it being within the control of the founder to leave a greater or less depth of more or

less tough grey cast iron to support that which has been chilled. This combination of chilled and ordinary grey cast iron is obtained by casting the metal in a mould having a cast-iron surface where the casting is to be chilled, the remainder of the mould being made of casting sand in the ordinary way. Rolls are generally cast so as to chill the surface to the depth of about half an inch. The head of large shot for artillery purposes is cast in chill to increase its penetrative power.

The rate of cooling modifies the properties of cast iron in another way, less within control. The more gradually the metal is allowed to cool, on solidifying the larger will be the crystalline structure developed, *i.e.* the coarser the grain ; from which it follows that the metal near the surface of a casting will be much finer in grain than that at the centre. In castings of small thickness this is a matter of but little moment, the cooling of the whole taking place with sufficient rapidity to ensure the difference in grain between the centre and the circumference being too slight to be of consequence. In large castings, where, owing to the thickness and the mass of metal, the centre must of necessity cool slowly, the effect of the difference in the crystalline structure is of considerable importance, as it materially affects the strength of the casting. Large solid castings in which great strength is required are now usually made in steel, cast iron being reserved for such purposes as light columns, which are made hollow, and pieces of machinery, which either have not to bear much strain or have not so great a thickness as to cause the grain to vary much.

There is another point in connection with the making of castings which does not generally receive the attention its importance deserves. When any substance crystallises, the crystallisation always takes place at right angles to the surfaces at which cooling or heating takes place ; in other words, the crystals will be formed so that their principal axes will be at right angles to the surfaces bounding the mass. The form



FIG. 33.



FIG. 36



FIG. 32.



FIG. 31.

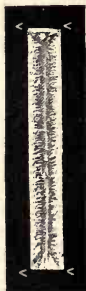


FIG. 35.



FIG. 34

in which iron tends to crystallise is a regular octahedron : an eight-sided figure, with equal axes at right angles to one another, a figure which may be represented by two square pyramids placed base to base. The importance of this subject in practice may best be shown by a few examples. If a square bar be cast, and cooled uniformly in all directions, there will of necessity be regions in which the crystallisation, proceeding uniformly from all sides, will meet. Along these planes of intersection (Fig. 32), there will be confused crystallisation, and lines of weakness in the casting will result. In a long rectangular bar the lines of weakness will be as shown in Fig. 35. Now let us observe what takes place in a round cast bar (Fig. 34). There are no lines of weakness. It is possible, then, by care in design to avoid many evils which might otherwise occur through ignorance of the laws of crystallisation. In Fig. 31 is represented a part of the lower portion of the first hydraulic cylinder made for raising the tubes of the Britannia Bridge. It broke under the great pressure to which it was exposed, the sides giving way along the planes of confused crystallisation, as in Fig. 36. A new cylinder was then made, the design being altered to that shown in Fig. 33. This answered perfectly, although the same kind of metal was used, and no increase of mass was made. This points to the simple conclusion that abrupt changes of form should be avoided.

The foregoing considerations apply equally when a mass of metal, which has been cooled without very marked crystallisation having taken place, is subjected to such conditions as will enable the ultimate particles of the metal to rearrange themselves, as when the metal is heated beyond a certain point, more particularly if, at the same time, it be subjected to vibratory motion.

The structure of malleable iron and soft steel is entirely altered by rolling, or drawing the metal out under the hammer : it becomes fibrous. The condition of the metal can readily be made apparent by placing it in dilute acid,

which will etch the surface in a manner corresponding to the structural arrangement. If a piece of malleable iron or steel be slowly bent until it breaks, the fractured surface will be fibrous, on the other hand, if it be nicked on one side and broken by a sudden blow, the fracture will be crystalline, showing that the fibre is developed by the drawing-out action during bending. Metal in which fibre has been developed becomes crystalline again when heated to a certain point. Vibration assists in converting fibrous metal back into a crystalline state when heated, a lower

temperature sufficing than in the absence of vibration; but experience has shown that vibration alone does not exert any marked influence on good metal, such as that employed for the axles of railway carriages, although inferior metal would be affected.

Remelting of Cast Iron in the Foundry.

—In order to melt the pig-iron for the production of castings, a *cupola furnace* is employed. The

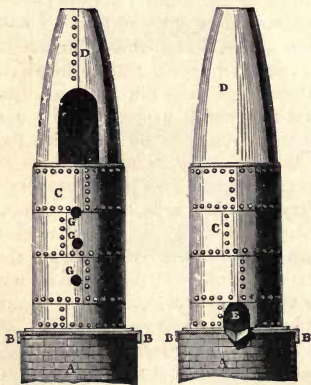


FIG. 37.—Front and Back Views of Cupola Furnace. A, Foundation of Masonry covered with Iron Plates B.

construction of this furnace varies in different foundries, but it is now commonly made of the form represented in Fig. 37, being cased with thick iron plates strongly riveted together, and protected internally by a layer of binding sand about nine inches thick. At different heights up the sides of the furnace there are openings (c) for

the introduction of blast-pipes, two or three of which are employed at one time, the remaining openings being closed by iron doors. For melting 5 tons of cast iron, the cupola furnace is 9 feet high, and $3\frac{1}{2}$ feet in diameter, clear of the lining. The tuyère holes, 6 inches in diameter, are placed at intervals of 15 inches, the lowest being 30 inches from the bottom, which is made to slope a little towards the gutter (E) by which the melted iron is run out. A conical iron hood (D) surmounts the surface, connecting it with the chimney, and having an opening for introducing the metal and fuel. A wood fire is first lighted in the cupola, through the tap-hole, then a quantity of coke having been thrown in, and well ignited by the blast, the tap-hole is closed, and the pig-iron introduced in pieces of about 28 lbs. each, together with one-fourth of its weight of coke. Fresh charges are introduced about every quarter of an hour, until the whole of the iron is melted, when the tap-hole is unstopped, and the metal run, either into moulds sunk into the floor upon which the furnace stands, or into a casting-ladle, whence it is transferred to the moulds.

During the process of remelting, about five or six parts of iron out of a hundred are wasted by combining with the oxygen of the air to form an oxide of iron, which unites with silica from the lining of the furnace to form a slag. The quality of the iron is generally improved in the process in consequence of the refining influence of the air, and a highly carburetted iron is employed, lest the fusibility of the cast iron should be diminished in consequence of the reduction in the proportion of carbon. It is necessary to employ coke as free from sulphur as possible, and a small quantity of lime is not unfrequently introduced with the iron, in order to combine with the silica contained in the coke, and prevent excessive waste of iron in the slag; but the lime is liable to corrode the lining of the furnace.

When casting in sand, it is necessary to prevent the direct contact of the melted metal with the sand, lest it should be

chilled ; for this purpose the *ironmoulders' blacking* is used, which is generally charred oak-wood ground to powder ; the gas evolved by this, under the action of the heated iron, suffices to prevent the contact of the metal with the mould.

For some purposes it is found advantageous to melt the foundry iron with about one-third of scrap wrought iron.

In order to protect castings from rust, they are brushed over with linseed oil, and suspended over a very smoky wood fire, being afterwards dipped into turpentine, when they acquire a bright, carbonaceous, protective coating.

When the grey samples of cast iron are acted upon by acids (diluted sulphuric or hydrochloric) the iron is dissolved, but the black particles of carbon are left, and these are found to possess the same properties as the natural variety of carbon known as black lead or *graphite*,* of which pencils are made. When the white cast iron is dissolved in acids, very little black residue of carbon is left, because the greater part of the carbon is present in the state of chemical combination with the iron, and, on being treated with acid, is evolved in the liquid and gaseous form.

About twenty years ago, 150 to 200 tons was considered a good yield ; now it is not uncommon for a single furnace to produce 500 and 600 tons of metal per week.

In the case of furnaces making hæmatite iron for conversion into Bessemer steel on the spot, the metal is now generally tapped into a large ladle, supported on a carriage drawn by a small engine, and taken straight to the *converter*, to be treated whilst still in a molten condition. This ladle, as it forms part of the Bessemer plant, rather than that of the blast-furnace, will be described later on. When the metal is not required for immediate use in the liquid form, it is tapped out of the furnace into moulds made in a bed of sand, forming what are known as *pigs* of metal. The moulds are formed by pressing long billets of wood, D-shape in section, into the sand. They are arranged in parallel

* From the Greek verb, *to write*.

rows, each furrow being connected at one end with a main channel, called the *sow*, by which the metal is fed to them from the furnace. When the metal has cooled to a certain point, the pigs are easily detached from the sow by prizing them up with a long iron bar. The pigs of metal are about 3 feet long and 4 inches in section, and weigh about 1 to $1\frac{1}{2}$ cwt. For particular purposes the metal may be cast in iron moulds ; this is less common than it used to be.

Methods by which Malleable Iron and Steel are produced from Pig-Iron.—The production of malleable iron in the blast-furnace depended, we have seen, not merely on the power of carbon to deprive the iron ore of its oxygen, but mainly on the decarburising action of oxide of iron. It is absolutely impossible in the blast-furnace to prevent the partially reduced ore from becoming more or less charged with carbon, owing to its power to decompose carbonic oxide. The only way to get over the difficulty is to leave sufficient ore unreduced, and then to heat the mixture, by which means both the carbon and the oxygen are removed in combination with one another.

Since, at a red heat oxygen and carbon have a greater affinity for one another than have iron and carbon, or than iron and oxygen, it follows that if oxygen, either combined with iron, or in the free state as it exists in the air, be brought in contact with pig-iron in a molten state, the carbon will be removed from the latter. When pig-iron became the principal product of the blast-furnace, the fact that it could be decarburised by means of atmospheric air was soon discovered.

The process was carried out in a 'bloomery'—a term derived from the Saxon *bloma*, a lump, having reference to the *bloom*, or lump, of iron resulting from the operation. The term *fnery* has also been applied to this furnace. It consists essentially of a shallow rectangular hearth, lined with iron plates 2 or 3 inches thick, protected on the inner side by a layer of 'brasque,' made of charcoal dust.

There are one or more holes in the front plate, through which to tap out the slag. Several twyers were sometimes used, more commonly only one, the inclination of which was varied from about 5° to 16° , according to the nature of the pig being treated. The depth of the hearth, measured from the nozzle of the twyer, varied between 6 and 10 inches in different localities, depending on the kind of pig-iron and fuel. Various improvements were made in course of time, such as the substitution of an iron bottom, cooled by water circulated through it, for a stone bottom, or one formed of cinder or small coal beaten down. The furnace in later times was enclosed, and the waste gases utilised for heating up the pig-iron ready for the next charge, thus economising time and saving fuel. Lastly, the blast was heated. This, however, was not done until about fifty years ago, and then only in a few localities.

Supposing the furnace to be hot and in order, charcoal is thrown into the hearth, and on it pieces of iron, 2 or 3 inches thick, which are covered with more charcoal, and the blast then let on. The metal slowly melts down, and as it drops to the bottom of the hearth it passes through the blast proceeding from the twyer, the result being that the silicon in the pig is more or less oxidised to silica, the carbon to carbonic acid, and the iron to oxide of iron. The oxide of iron combines with the silica—besides that derived from the silicon in the iron, a good deal is introduced adhering mechanically to the pigs, in consequence of their being cast in sand—the liquid slag thus formed dissolves a further quantity of oxide, and being intimately mixed with the carburised iron, reacts upon it, removing the carbon. Rich oxide, remaining in the hearth from the previous operation, is also pressed into service. The now pasty mass is constantly broken up and lifted in front of the twyer, in order to expose the still unfined portion, or ‘raw’ iron, to the action of the blast. When the metal is considered sufficiently fined, it is removed from the hearth in the form of a spongy

mass, or *ball*, charged with slag. Being still at a white heat, it is placed under the hammer, by which means the slag is squeezed out, and the iron welded into a rectangular slab or bloom. The operation requires in all about one hour and a half. The slag, which contains 70 to 80 per cent. of protoxide of iron, is returned to the finery. The bloom is subsequently reheated, and drawn out under the hammer into a bar.

White pig-iron is better suited than grey for conversion into malleable iron, owing to its becoming pasty, in which condition it can be thoroughly incorporated with the cinder rich in oxide of iron. The grey iron is not so well suited, for it passes directly from the solid to the liquid state, as already explained, and the cinder then simply floats on the top, exerting only a very modified deoxidising action.

The removal of phosphorus is effected in the finery in exactly the same way that it is in the blast-furnace, when making malleable iron, by the action of excess of oxide of iron on the carburised metal. The slag, which is highly 'basic,' *i.e.* contains much more oxide of iron than is required to saturate the silica, first effects the removal of the silicon, then the carbon, and lastly the phosphorus. It is essential that the slag should be basic, for in the presence of free silica, or of an 'acid' silicate—*i.e.* one in which the affinity of the silica for a base is not fully satisfied—the phosphorus would not be removed from the metal, the silica having a greater affinity than phosphoric acid for oxide of iron.

When it was desired to make grey pig iron into malleable iron, it was first converted into white iron by treating it in a 'refinery.' The Germans call this process 'whitening' (*weiss machen*)—a better term than refining, which is misleading, since it would imply that the product is more highly purified than that attained by fining, which is not the case.

Besides converting the grey iron into white iron, this

process removes the greater part of the silicon, thereby facilitating subsequent operations.

The most modern form of refinery consists of a rectan-

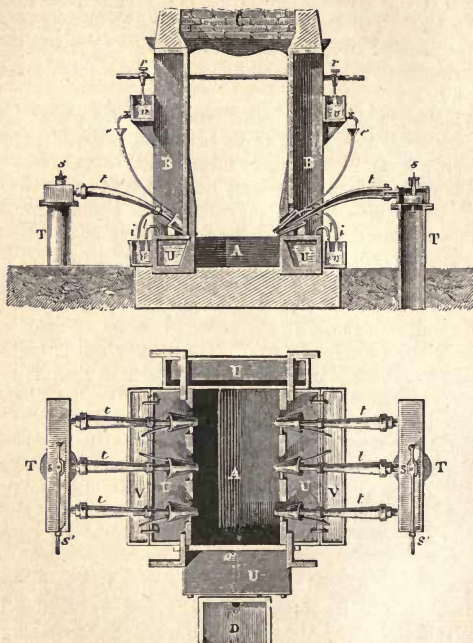


FIG. 38.—Hearth for refining Cast Iron, in section and plan. A, Hearth in which the fuel and pig-iron are placed. B, Hollow iron supports for the chimney C, which is about 18 feet high. D, Flat mould for the refined iron. e, Funnel-pipes for conveying water to the tuyères. i, Pipes for carrying off the heated water from the tuyères. l, Pipes for carrying off the heated water from the hollow walls U. o, Tap-hole for running out the fine metal. r, Stopcocks for supplying water to the cisterns v. s, Valves for regulating the blast issuing from the pipes T.

gular hearth (A), about 3 feet long, 2 feet wide, and $2\frac{1}{2}$ feet deep, enclosed on three sides by water-cooled iron blocks (U), and in the front by a solid cast-iron plate, having a tap-hole at the bottom of it. The twyers (T) are inclined at an angle of 25° to 30° to the horizontal. About 400 cubic feet of air per minute are supplied to the hearth, at a pressure of from $1\frac{1}{2}$ to 3 lbs. per square inch. The number and position of the twyers varies in different localities.

The hearth is partially filled with coke, which is the fuel generally employed ; on it the pig-iron is piled, and more coke having been placed on the top, the blast is let on. In less than an hour and a half the charge is melted down ; it is, however, kept under the blast for about half an hour longer, more or less, according to the degree of purity of the metal, and the extent to which it is desired to refine it. The *fine metal*, or *plate metal*, and cinder are tapped out into a shallow cast-iron mould (D), lined with loam, and kept cool by water circulated through it. The metal is thus cast into a plate about 10 feet long, 3 feet wide, and 2 inches thick, whilst the bulk of the slag runs off into a separate mould beyond, the remainder being easily detached from the surface of the metal. To assist the chilling, water is thrown over the metal, which can then be readily broken up into pieces of the required size. The metal should be free from honeycomb, and the fracture bright.

Notwithstanding that the metal is kept covered with coke or charcoal, owing to the inclination of the twyers and the pressure of the blast, the process is a strongly oxidising one.

Practically but little phosphorus is removed in this process ; a small quantity does, however, pass into the slag, notwithstanding that the carbon in the metal has only been reduced in quantity very slightly. The reason of this is that the proportion of silicon, carbon, and phosphorus present being very small in relation to the iron, they are too diffused to exert their individual influence throughout the mass. It thus happens that the carbon begins to oxidise

before all the silicon has been removed, and, similarly, the phosphorus to some extent is affected, whilst carbon is still present. This is well shown in the case of the iron, which, although less oxidisable than either the silicon, the carbon, or the phosphorus, yet, owing to the greater surface exposed, is largely oxidised before these other bodies are removed. They are, in fact, removed by their depriving a part of the oxide of iron of its oxygen.

Sometimes the metal was run straight from the blast-furnace into the refinery. By this means a considerable saving in fuel was effected.

When the metal is intended for making sheet iron for the best tin-plates, charcoal is used instead of coke, both in the refinery and the finery, in order to avoid introducing sulphur. The commoner plates are made from iron which has been refined with coke.

*The metal from the finery, when intended for the best plates, undergoes a selecting process. The ball is shingled, and drawn into a bar $1\frac{1}{2}$ to $2\frac{1}{2}$ inches thick, which is then broken into lengths, called *stamps*, weighing about $\frac{1}{4}$ cwt. each. The stamps are carefully sorted, those which do not show a fine, bright, and uniformly crystalline grain being rejected, and returned to the finery with a subsequent charge. The selected stamps are made into a pile, reheated in a *hollow-fire*, and rolled down into sheets.

The hollow-fire consists of a deep rectangular hearth, the lowest part of which contains coke, kept at a strong heat by means of a blast from inclined twyers. The pile, supported on a *staff*—a long bar with the end flattened out—is held in the flame above the coke, so as to avoid contamination with sulphur by direct contact with the fuel.

The manufacture of charcoal- and coke-plates by this method has hitherto formed an important industry in South Wales; but it has now very generally given place to the Siemens process, by means of which plates can be produced superior in quality even to the best charcoal-plates.

By slightly modifying the process, steel can be produced in the finery instead of malleable iron, just as in the case of the Catalan forge and the primitive blast-furnaces. The method of conducting the finery process, for either iron or steel, will vary somewhat according to the kind of pig-iron and the locality in which it is practised. Each district has its own ideas as to the proper way of arriving at the best result, and besides the depth of the hearth, the slope of the sides and the inclination of the twyers have to be modified according to the character of the metal to be treated.

Broadly speaking, the chief difference between an iron finery and a steel finery is that the depth of the latter is less, and the twyer both lower and more inclined. Steel and malleable iron are, however, sometimes produced alternately in the same hearth without difficulty.

When it is intended to make steel, the metal is not broken up after it has been melted down, but is kept covered with a layer of liquid slag. The process is conducted much more slowly, requiring about half as much time and twice as much fuel as in making malleable iron. The blast is used at lower pressure. The purest pig-iron and fuel alone should be used, as impurities affect the quality of steel much more than they do iron. This process has been largely used in Styria, Carinthia, and the Tyrol, very high class steel being produced.

In the Siegen district a piece of pig-iron weighing from 50 lbs. to 60 lbs. is placed on the hearth, having been previously heated; the hearth is then three-quarters filled with burning charcoal; on it is placed a portion of the cake produced in the last operation, and which has been kept hot in burning charcoal at the back of the furnace. The remainder of the hearth is then filled up with charcoal. The other six or seven pieces, into which the last cake was divided, are placed on the top. In this process the production of steel, and the reheating of that obtained in the last operation preparatory to working it under the hammer,

are conducted together. The blast is turned on. The piece of pig-iron forms into a pasty mass; cinder, rich in oxide of iron, produced in the latter part of the preceding operation, is then thrown in; a second piece of pig-iron, weighing about 100 lbs., is added, and afterwards four or five pieces of spiegel-eisen, weighing each about 100 lbs., are successively added. The spiegel-eisen contains about four per cent. of manganese. If the metal be found to be too much decarburised, more spiegel is added. The cinder is usually allowed to rise 2 or 3 inches above the cake of metal, any excess being tapped off. There are several modifications of this process, but enough has been said to make the principle clear. In this, as in the Catalan process, it is impossible to obtain a homogeneous product, except in small quantity. The principle in both is essentially the same, viz. decarburisation by oxide of iron. In the finery process, in addition to the oxide added in the form of cinder, and scale produced during the working of the metal under the hammer, some results from the reheating which we have seen is carried on at the same time. In this process manganese also plays an important part, and we shall see that in every process for the production of steel manganese is used with great advantage.

All the processes which have been referred to in the foregoing pages are still in use in some parts of the world, though circumstances may have led to their being discarded in others, in which at one time they were in use. It thus happened that in England the finery process gave way to the *puddling* process. This was brought about by the increasing scarcity of wood for making charcoal. We have seen how coal, after being coked, came to be substituted for charcoal in the blast-furnaces of this country, though charcoal still holds its own where it can be obtained in sufficient quantity. In course of time, for the same reason, it was attempted to employ coal instead of charcoal in the manufacture of malleable iron. Neither coal nor coke could be

used in the finery with satisfactory results, owing to the sulphur they contain. It was, therefore, sought to utilise the coal by burning it in a separate chamber, and only allowing the flame to come in contact with the metal being operated upon. In this way the sulphur in the fuel is converted into sulphurous acid gas before coming into contact

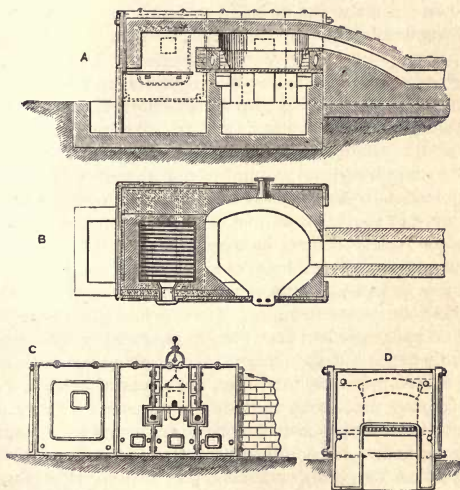


FIG. 39.

with the iron, upon which it then has no action, the affinity of the sulphur and oxygen for one another, when once chemically combined, being greater than that exerted between the combined sulphur and the uncombined iron.

The idea of manufacturing malleable iron in this way first began to take shape early in the eighteenth century ; it was

not, however, till towards the close of that century, less than a hundred years ago, that it was actually carried into practice.

The most modern form of the furnace, in which the operation known as puddling is carried out, consists of a rectangular fire-place, a hearth or bed, and a flue connected with a stack or chimney. The hearth is separated from the fire-place and the flue by bridges, known as the fire-bridge and the flue-bridge respectively. The grate is formed by supporting wrought-iron bars on transverse bearers. The depth and area of the fire-place depends on the nature of the fuel employed, being least with highly bituminous coal, and greatest with anthracite. A dry, non-caking coal, burning with a long flame, is that to be preferred. The fuel is introduced through an opening about 10 inches above the grate, which is kept closed with a few shovelful of coal when not in use. The bottom of the earth is constructed of cast-iron plates, about $1\frac{1}{2}$ inch thick, rabbeted together, and supported underneath by bars of wrought iron, resting on cast-iron standards, or some similar simple arrangement. The working bottom is formed by coating the iron plates with substances rich in oxide of iron, the slag produced in the furnace towards the end of the process itself, hammer scale, *i.e.* the slag and scale squeezed out from the bloom of metal under the hammer, &c., being used for the purpose. These substances are strongly heated in the furnace, and then spread over the bed to a depth of 2 inches by means of a flat bar, or *paddle*. The hearth is enclosed by iron plates or hollow castings, backed with fire-bricks. These side plates are protected on the inner side in a similar way to the bottom. The material commonly used for lining or *fettling* is termed *bull-dog*; it is obtained by roasting *tap-cinder*—the slag of the puddling-furnace. Tap-cinder is essentially a silicate of protoxide of iron. ($2\text{FeO}, \text{SiO}_2$). When this is strongly heated in the presence of air, the protoxide becomes converted into peroxide (Fe_2O_3), which has no affinity for

silica ; so that, instead of an easily fusible chemical compound, a mechanical mixture of two highly infusible substances is obtained. The layer of bull-dog is coated, so as to produce a smooth surface, with *puddlers' mine*—a soft unctuous variety of red hæmatite, worked into a paste with water. The fettling requires to be repaired after each heat, and after each shift of twelve hours the bottom is made up by throwing in about 1 cwt. of scrap iron, which is allowed to oxidise, and is then worked over the bed.

In the early puddling-furnaces sand bottoms were used. Owing to the great affinity of silica for oxide of iron, much waste of iron resulted from their use, and the elimination of phosphorus was rendered much more difficult. The introduction, at the beginning of this century, of the plan of making the bottom and sides of iron plates, and fettling in the way just described, was a great step in advance, and the weekly yield of the puddling-furnaces is said to have increased from 8 tons to 24 tons in consequence, mainly owing to the decreased wear and the greater facility for making repairs.

The fire and flue bridges are usually made hollow, so that they may be kept cool by circulating air or water through them. The sides of the hearth are sometimes also kept cool in the same way.

The working-door is generally made of fire-brick slabs, held in a cast-iron frame, suspended from a counterpoised lever, the lower edge being about 10 inches from the bed. At the bottom of the door is a notch called the *stopper-hole*, through which the tool employed in stirring or *rabbling* is introduced, in order to avoid admitting too much air, and cooling the charge by opening the door more than absolutely necessary. Immediately beneath the working-door is the tap-hole, through which the slag or tap-cinder is run out from the hearth.

The roof, which is made of fire-brick, forms a low flat

arch, sloping slightly towards the flue. The sectional area of the flue varies according to the nature of the fuel employed, being greatest for highly bituminous, and least for anthracitic coal. The stack, with which the flue is connected, is usually from 30 to 50 feet high when drawing only one furnace, and about 20 inches square when the waste heat is not utilised for raising steam. If the stack serves more than one furnace, or the spare heat is utilised, it might be necessary to increase the height to 100 feet or more. There is no advantage, beyond a certain point, in attempting to apply the waste heat to raising steam, as it is absolutely essential that a considerable quantity should pass up the stack, in order to create the necessary draught. The smaller the amount of heat allowed to escape, the higher must the stack be made. The draught is sometimes assisted by closing in the underneath part of the grate, and forcing in a blast of air to increase the intensity of combustion. The regulation of the draught is effected by means of a damper placed at the top of the stack. On the proper use of the damper at different stages of the process depends, in a great measure, the success of the operation.

There are only two kinds of tools employed in ordinary puddling: the *rabble*, and the *paddle*. The former consists of an iron bar about 8 feet long and $1\frac{1}{4}$ inch diameter, the working end being turned at right angles, and rather broader. The latter is a somewhat lighter bar, made at the end to a chisel shape. These tools are not allowed to remain in the furnace more than a few minutes, otherwise they would soften, and stick to the metal being worked. When removed they are immediately plunged into water.

Siemens' regenerative system has been applied to puddling-furnaces, but the number of works where it is in use are very few.

There are two methods of conducting the puddling process: the one called 'dry' puddling, the other 'pig-boiling.' The former is the original process, the latter being that

almost universally employed. For dry puddling, the iron must be white, or have been previously refined, so that on heating it it may become pasty, and the oxide formed by exposure to the action of the air can then be incorporated with the metal, by which means the carbon, &c. are oxidised. The metal, which is never allowed to become fluid, is constantly broken up, and the oxide of iron worked in with it. When sand bottoms were given up, this method of puddling was performed on an iron bottom, thinly coated with cinder. There is less waste of metal and a smaller consumption of fuel in this process than in the boiling, but the quality of the iron produced from the same class of pig, if not the best, would not be so good, the conditions of the operation not being so favourable to the removal of phosphorus and sulphur.

In pig-boiling, grey iron is used, which forms a fluid bath, covered with a layer of slag, which protects it from the action of the air, and the decarburisation of the metal is therefore effected, practically, entirely by means of oxide of iron derived from the fettling, and from scale forming part of the charge. The only occasions when the air plays any part in the decarburising are during the melting down of the pigs, and after the metal has given up most of its carbon, when it becomes thick and pasty, and is said to 'come to nature.'

The charge of pig and hammer-slag, &c. having been thrown into the furnace, the damper is raised, and if a blast is employed, it is turned on. In about thirty minutes the metal has gradually melted down, and become covered with a thin layer of cinder. The temperature of the furnace is now sufficient to fuse the hammer-slag, which, by reason of its less specific gravity, rises slowly through the metal, exerting a powerful decarburising effect. The damper is now shut down, the blast turned off, and intimate contact between the molten oxide and the metal is promoted by stirring the charge with the rabble, worked with short quick strokes. It

is advantageous to check the rise of the temperature at this point, otherwise the charge would become too liquid, and the cinder rise rapidly through the metal without exerting its full decarburising power. In about forty minutes from the commencement violent action sets in, the surface of the molten mass presenting the appearance of boiling, being agitated by the escape of carbonic oxide, formed by the union of the oxygen from the cinder with the carbon of the pig. The gas as it issues burns in small jets, known in the forges as puddlers' candles, having the characteristic blue colour of carbonic oxide undergoing combustion. During this stage, which lasts about a quarter of an hour, the metal, which has been rabbled vigorously all the time by the puddler, becomes less and less fluid, ultimately, when the carbon has been reduced to about 1 per cent., becoming infusible at the temperature of the furnace. The temperature is now raised, and the pasty mass of metal and slag broken up repeatedly. During this period the greater part of the remainder of the carbon is removed, and, finally, the phosphorus. When the puddler considers the iron to have been brought to a satisfactory condition, he reduces the temperature, and works the contents of the furnace up into balls of about seventy pounds weight, which are then raised to a welding heat, and then taken from the furnace straight to the squeezer or hammer, where the spongy metal is consolidated into a slab or bloom, the cinder being squeezed out. The puddled iron is never perfectly freed from carbon, the quantity remaining in varying up to about 0.25 per cent. To convert the charge of pig-iron into malleable iron by this process takes about one hour and a half, more or less, according to the character of the pig. Each complete operation is termed a *heat*. When pig containing an appreciable quantity of sulphur is being used, some manganese ore in the fettling is beneficial.

Where the manufacture of iron is carried on in accordance with the most modern practice, steam-hammers alone are

employed, the use of squeezers and tilt- and helve-hammers being confined to countries in which less progress has been made. In single-acting hammers the steam is only employed to lift the head, which is then allowed to fall by its own weight. In double-acting hammers the steam is admitted alternately on each side of the piston, thus increasing the force of the blow which the hammer is capable of striking. These hammers have a great advantage over the old-fashioned helve- and tilt-hammers, inasmuch as the force of the blow can be regulated to a nicety by checking the exit of the exhaust steam, and can be increased, or decreased from moment to moment at the will of the operator. Thus, in treating a puddled ball, the blows at first should be light, and gradually increase in force as the mass consolidates. If heavy blows were applied at first the ball would break to pieces, whilst later on light blows would be equally objectionable, since they would fail to effect the object of the operation, viz. to render the metal as homogeneous as practicable, by squeezing out as much of the slag as possible. These hammers are made of all sizes, according to the nature of the work ; hammers varying in weight from 30 cwt. up to 80 tons, and having a drop of 16 feet, have been employed. Steam is not generally used above the piston in hammers weighing more than 12 to 15 tons.

The shingled bloom, being still at a high temperature, is passed to the puddle-rolls, technically called the *forge-train* or *mill*, which consists of two pairs of rolls, the one (A A'), for *roughing down*, the other (B B'), for finishing. The roughing-rolls have on their surface angular grooves, which gradually diminish in depth from one end of the series to the other. The grooves in the finishing-rolls are rectangular, and also form a diminishing series. After passing through each groove the slab or bar is lifted on to the top roll, which, as it revolves, carries it back into position to pass through the next. In order to prevent the delay thus occasioned, either reversing engines or three high rolls are now

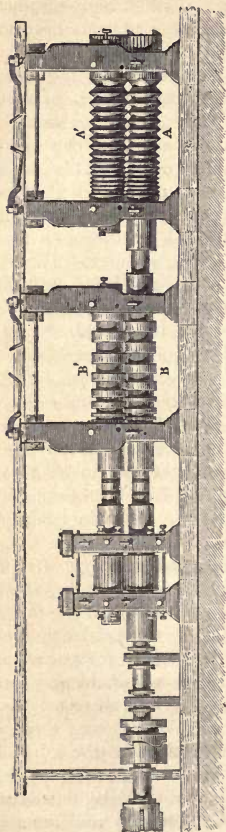


FIG. 40.

commonly used (Fig. 41), so that the metal is passed through the rolls first in one direction and then in the other, alternately. After it has passed once through the series of grooves in the roughing and finishing rolls, it is classed as No. 1 iron, or puddled bar. It will readily be understood that the iron thus produced will be far from homogeneous, being more impure in some parts than in others, and containing more or less mechanically admixed slag. In order to obtain a better class of iron, the No. 1 bar is cut into lengths, *piled*, reheated, and again rolled, forming No. 2 iron, which, if a better quality be required, may be similarly treated, one or more times, to produce No. 3, or a higher quality. The shape of the pile will depend on whether bars, rails, angle-pieces, &c. are being manufactured. Sometimes a different quality of iron is placed in the interior of the pile to that used for the exterior. Thus a metal of high tensile strength, but

somewhat brittle, might be placed in the centre of a pile intended for a rail, the exterior being composed of a tougher metal. The pieces composing a pile are tied together with wire, to keep them together until they have become sufficiently heated to adhere to one another. Very frequently the welding of a pile is effected partly under the hammer, and completed in the rolls, or the pile, before going to the forge-train, is put through a blooming-mill, consisting of a set of two pairs of horizontal rolls, with a pair of vertical-grooved rolls running between them.

The *reheating* or *mill furnace*, in which the piles are brought to a welding-heat, in general appearance resembles the puddling-furnace, with the exception that the grate is smaller in proportion to the hearth, and the bed slopes towards the flue, where there is a tap-hole, through which the cinder formed during the reheating runs off. The bottom is constructed of fire-brick, coated with sand, rammed

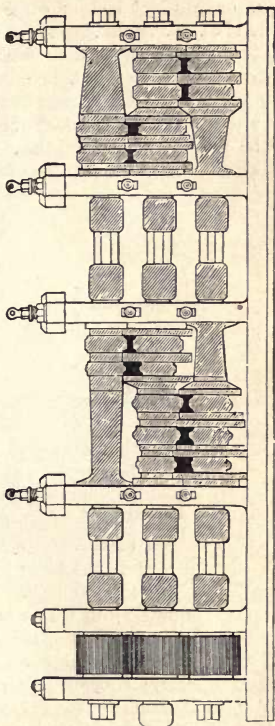


FIG. 41.

in while moist. When the pile cannot be rolled straight down without reheating, the first heating is sometimes given on a 'dry' bed of oxide of iron, to lessen the waste ; but, in any case, before the completion of the rolling of the bar or rail, it must receive a 'wash' heat, *i.e.* must be heated on a sand bed, in order that any oxide of iron may be dissolved out, leaving clean metallic surfaces, without which the pieces of which the whole is built up could not be properly welded together. For the same reason a blacksmith, when welding two pieces of iron, sprinkles sand

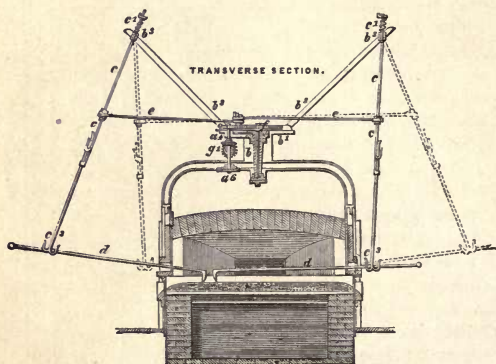


FIG. 42.

over the surfaces to be united. The sand forms a fluid silicate ($2\text{FeO}, \text{SiO}_2$) with the oxide on the surfaces of the metal, protecting them from the further action of the air, and being easily squeezed out when the two pieces are pressed together, producing true metallic contact, a condition which is absolutely essential to obtain a good weld.

Reheating-furnaces, worked with gas on the Siemens system, with or without regenerators, are largely used.

During the reheating, every precaution possible to exclude

air from the hearth should be taken, the doors being well closed, and care being taken to maintain a reducing atmosphere, which latter condition is best fulfilled when gas firing is employed.

Mechanical Puddling.—The labour involved in ordinary puddling is very great, requiring considerable physical strength and power of endurance. Many attempts have been made to perform the work of the puddler by machinery, but for various reasons none of the numerous arrangements

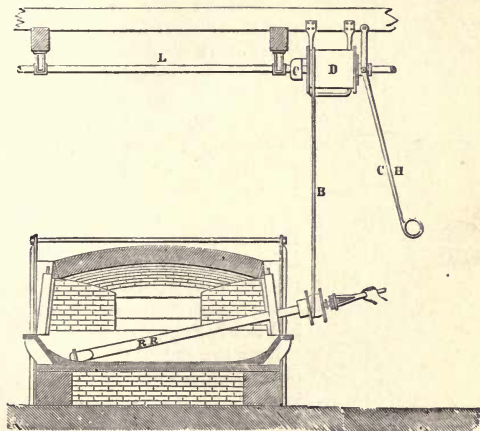


FIG. 43.

which have been tried have remained in general use. There are only a very few works at which mechanical puddling is carried on at the present time. Machine puddling may be divided into two classes :

I. In which ordinary hand-puddling is imitated, the machine doing the greater part of the work, under the guidance of the puddler, who finishes the heat in the usual way by hand.

2. In which manual labour is dispensed with altogether, the ordinary process being reversed, rotary motion being imparted to the furnace, thereby bringing the charge to 'nature' without the aid of tools.

Figs. 42, 43 and 44 will sufficiently indicate the nature of the appliances referred to under head No. 1. Fig. 42 represents Witham's puddling machine. The inventor has had

a number of these machines in successful operation at his works since 1865; there are also several of them in use at Woolwich Arsenal. The cut is a cross-section of the furnace. It will be observed that the furnace is worked from both sides; in other respects it does not materially differ from an ordinary puddling furnace. Figs. 43 and 44 illustrate a different form of rabble. Instead of being made to traverse the bed with a reciprocating motion, as in the type of machine just considered, it is revolved by the strap B, which causes it to move along the bed in a horizontal direction.

Near the end of the rabble is a disk, which is replaced by a spiral when the metal is coming to 'nature.' The spiral is seen in Fig. 44, which is an enlarged section of the rabble,

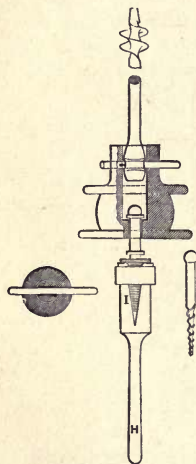


FIG. 44.

arranged to show its construction.

Of the machines coming under head No. 2, the first to meet with any real success was that invented by Danks (Fig. 45). A large number of these machines and various modifications of it have been tried in England, but at the present time there is not one in use in this country. In America there are a few of these furnaces. A blast is

supplied under the grate by the *wind-pipe*, and a fan-blast is also introduced above the grate. By controlling the

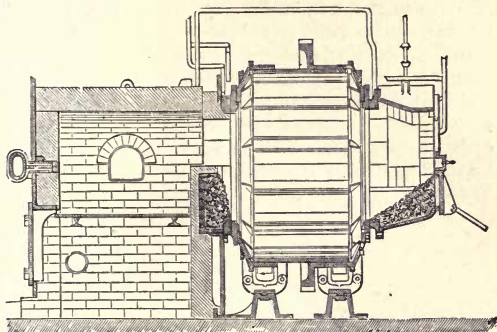


FIG. 45.

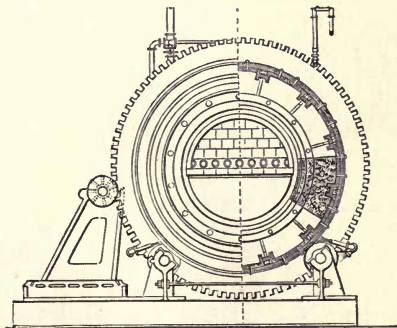


FIG. 46.

quantity of blast introduced, the temperature is regulated according to the requirements of the process at its different stages. The bridge-plate separating the fuel-chamber from

the rotator is cooled by water circulating through a coil. Attached to the bridge-plate is a water-cooled ring with a flat outer surface, cast in chill; against this one end of the rotator presses, forming a butt joint. The rotator is a cylindrical chamber, constructed externally of iron, and lined internally with refractory material. One end, as already described, communicates with the grate; the other has a double purpose, serving for the introduction and withdrawal of the charge, and as an exit for the products of combustion, for which purpose, during the puddling operation, it is con-

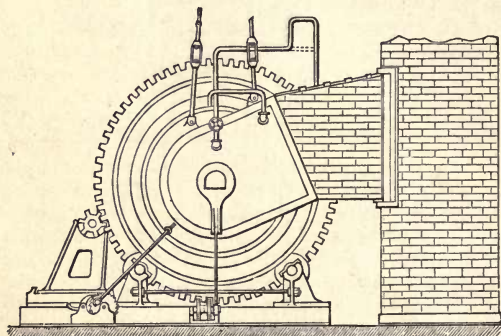


FIG. 47.

nected with a chimney or stack by a movable flue, worked by overhead gearing. The revolving chamber is supported on ordinary friction rollers, and rotated at whatever speed may be required by suitable machinery. Wherever necessary water-cooling is resorted to. The metal casing, which has wedge-shaped recesses in the inner side, is first coated with a mixture of pulverised iron ore and cream of lime, made into a mortar, which soon sets, and the furnace is then ready for fettling. To effect this, iron ore, as free as possible from silica, is thrown in and melted, the furnace being

heated up and rotated. Into the pool of melted ore, lumps of ore, of such dimensions as to project above the surface some inches, are thrown, and these being cold, the melted ore sets and fixes the lumps. By repeating this operation on successive portions of the initial lining, the interior of the furnace is gradually fettled throughout.

The furnace may be charged either with solid pigs of iron, or with liquid metal direct from the blast-furnace. About half an hour is required to melt a charge of solid pig. When the whole of the charge is in a molten condition, the furnace is revolved slowly for five or ten minutes, in order to bring as much as possible of the metal in contact with the fettling. With the object of further promoting the action of the cinder on the carburised iron, a stream of water is injected through the stopper-hole on to the lining of the furnace on the descending side, just above the liquid cinder floating on the metal. This causes some of the cinder to solidify, and to be carried down continuously beneath the surface of the metal, where, becoming melted again, it rises up to the surface, exerting in its passage a powerful decarburising action. At the expiration of five or ten minutes the metal begins to thicken, the rotation is stopped, and the heat raised, to liquefy as much as possible the cinder, which is then run out by bringing the tap-hole just above the surface of the metal. The tap-hole having been plugged up again, the temperature, which has somewhat dropped, is again raised, and the furnace set in motion at about six or eight revolutions per minute. During this period the remainder of the carbon and impurities are removed, and, in order to collect the charge into a ball, it is only necessary to decrease the speed to two or three revolutions per minute. The ball is got out by removing the flue, inserting a large fork slung to a crane, and giving the chamber a partial rotation to cause the ball to roll on to the fork, which is then removed by means of the crane, and the ball carried off to the squeezers.

The principal difficulties with this type of puddling machine are to get a cheap fettling, which will stand the wear and tear, and workmen who will take the trouble to keep the apparatus, as a whole, in proper order. There appears no reason to doubt that, with proper management, very excellent iron can be produced in these furnaces, even from highly phosphoretic ore ; but, from an economical point of view, judging from the fact that none are now in use in this country, it may safely be concluded that they have not so far been successful. This want of success is, it is to be feared, in no small measure due to those who have had the handling of these machines, rather than to any innate fault in the apparatus itself. Puddling is still, as it has been from the first, the apotheosis of brute force, any attempts to give it refinement being ignominiously hurried out of existence. But science is now having its revenge, or, more properly speaking, its triumph. With slow but certain step, during the last twenty years, it has been displacing this crude, brutalising process. Its death-knell has been sounded.

With one notable exception—the cementation process—the early methods for the production of steel were simply modifications of the methods for producing malleable iron. Accordingly, we find that the introduction of the puddling process, by which malleable iron is produced in a reverberatory furnace, was soon followed by a similar process for the manufacture of steel.

In order to produce steel, instead of being totally deprived of carbon, as in the manufacture of malleable iron, the metal is only partially decarburised. More time and labour are required to bring about anything like a satisfactory result than in making iron. The steel being fusible at a lower temperature than malleable iron, it has to be balled and welded at a lower heat. This makes it more difficult to squeeze out the slag, which tends to thicken at the lower temperature. For this reason it is essential that manganese should form a constituent of the charge, as thereby the

slag is rendered sufficiently fluid, and also exerts a less decarburising action. It is very important that the slag should be removed from the spongy ball of steel as rapidly as possible. Even when charges of only 2 or 3 cwt. are worked at a time, it is practically impossible to produce a steel of uniform composition. This steel also has the drawback that it welds badly, probably owing to the impossibility of removing all the cinder. This difficulty, which is common to all steel which has not undergone fusion, has, however, been got over by completely fusing the steel before working it, so as to enable the slag to completely separate.

The same principle as that which regulates the production of steel by the foregoing methods was taken advantage of in the Uchatius process, which was patented in 1855.

Pig-iron is first granulated by running it, while molten, into cold water. The granulated metal is then mixed with about 20 per cent of roasted spathic ore, crushed fine ; the mixture, to which a little flux has been added, if necessary, is then fused in clay crucibles. If very soft steel be required, some wrought-iron scrap is added.

By all the processes we have so far reviewed good steel could be produced, but only in comparatively small quantity, and at great expense. The applications of steel were in consequence very limited ; in fact, practically its use was confined to implements with a cutting edge.

Occasionally masses of steel weighing many tons were produced by melting the metal in a vast number of crucibles, the contents of which were poured simultaneously into a mould. This plan, although effective, was extremely laborious and expensive.

In 1845 Heath patented a process which, had it been successful, would have given him the power of producing steel in quantity. He proposed to melt scrap iron in a bath of molten pig-iron, in a reverberatory furnace heated by jets of gas. There were two conditions wanting in this method, which caused it to be a failure, viz., a sufficiently high

temperature and the power to easily regulate the character of the gases employed. Nevertheless, in this suggestion is to be found the germ of one of the two most important processes of the present day. By the foregoing remarks it is not intended to imply that the idea of mixing wrought and cast iron together to produce steel was originated by Heath. On the contrary, as we should expect, the idea was a very old one : in 1722, Reaumur tells us that he succeeded in making good steel in a common forge in this way. It would appear, however, that Heath was the first to suggest the use of a reverberatory furnace and gas for the purpose, and that is the important point.

By the introduction, in the year 1856, of the regenerative system of heating,* the only remaining obstacle to the production of steel on a large scale by means of reverberatory furnaces was removed.

It is at once apparent that we have here the very conditions which were wanting to make successful the process patented by Heath in 1845, for not only have we the high temperature, which could not then be obtained, but it has become easy to create at will, by regulating the relative proportions of combustible gas and air, either an oxidising, a reducing, or a neutral flame.

There are two methods now in use for the production of steel in the reverberatory furnace, or open hearth, as it is called. In France, pig-iron and scrap steel are fused together ; in England, pig-iron is decarburised by means of iron ore, some scrap, however, being generally added, for the sake of utilising it. The necessary amount of carbon is imparted to the metal by means of spiegel-eisen or ferro-manganese.

This process has been largely employed for the production of ship- and boiler-plates. It has the great advantage that the metal can be kept fluid on the hearth, and its composition adjusted until it is exactly that required.

* See article on Fuel, p. 63

In the Siemens-Martin or pig-iron and scrap process, the necessary decarburisation is effected by means of a sufficiently oxidising flame, whilst in the Siemens, or pig and ore, process the decarburisation is brought about almost entirely by the ore added, about half of which becomes reduced, thereby increasing the make of metal by 1 or 2 per cent., the rest remaining in the slag. The ore used must be free from phosphorus and sulphur. Somorostro or Mockta ores are preferred, but Marbella, Whitehaven, and Cornish are sometimes used. About 1 ton of ore is required for a 5 ton charge. In either process the carbon is gradually reduced until about only 0·10 per cent., or less, remains in, when the requisite amount of spiegel-eisen or ferro-manganese, or a mixture of both, is added. In some works the spiegel is added to the charge in the furnace just before tapping, and the ferro is sprinkled on to the metal as it flows into the ladle, it being maintained by those who pursue this practice that a perfect mixture is obtained in this way, whilst less waste of manganese results. That a homogeneous metal is invariably obtained in this way is, in the author's opinion, more than doubtful, and the practice of adding the whole of the manganese before tapping is strongly to be commended, notwithstanding that it may entail a somewhat greater expense; for it must be borne in mind that the failure of a single plate in a boiler may have the most disastrous consequences, causing serious prejudice in the mind of the public against a metal which, when carefully manufactured, is vastly superior to even the very best qualities of wrought iron.

At first sight it will not be apparent why a metal of the desired character should not be produced by simply reducing the carbon to the necessary extent, and then tapping. This cannot, however, in practice be accomplished, it being found that under these circumstances a 'short' worthless metal is obtained. This difficulty can be got over by adding manganese; but, then, manganese in the metallic state, uncombined with any other substance, is comparatively very

expensive, whilst, in the form of spiegel-eisen, or ferro-manganese, combined with iron and carbon, it is relatively cheap, so that it is far more economical in practice to wholly decarburise the bath of metal, and then impart to it the requisite amount of carbon by means of ferro or spiegel. This plan further recommends itself on the score of simplicity, it being much easier to entirely decarburise, and then add a certain amount of spiegel or ferro, containing a known quantity of carbon, than to determine the exact point at which to stop the decarburisation. In a few exceptional cases, in which the pig contains two or three per cent. of manganese, and is free from sulphur, it is, however, practicable to get sound metal without the final addition of manganese, provided a hard metal, high in carbon, is required. This can in no case be done in the ore process, as the manganese, having a greater affinity than iron for oxygen, rapidly replaces the latter in the slag, so that none is left in the metal at the time it would be required to exert its action. The fact that the metal, before the addition of manganese, is red-short* is commonly attributed to the presence of oxygen in combination with iron, either mechanically mixed with, or in solution in the mass of the metal. The probability of this is not difficult to realise, when it is remembered that the decarburisation is effected by oxygen, that iron has a great affinity for oxygen, and that at a certain stage of the process the carbon is reduced to a very small amount. This view is supported by the fact that only a portion of the manganese and carbon, introduced as spiegel or ferro, is subsequently found in the metal, even though the addition may have been made in the ladle after tapping. This view is further corroborated by analysis, which has shown the presence of 0.35 per cent. of oxygen in metal containing 0.08 per cent. of carbon. Our faith in this solution of the problem as universally applicable is, however, shaken, on considering harder metal, containing from $\frac{1}{4}$ to $\frac{1}{2}$ per cent.

* Brittle at a red-heat.

carbon. The evidence is against the metal, under these circumstances, containing oxygen, for, on adding spiegel, it remains tranquil, *i.e.* there is no evolution of gas, which would take place were oxygen present. Yet, if the metal be simply decarburised, until it contains from $\frac{1}{2}$ to $\frac{1}{4}$ per cent., and manganese be absent, it will be red-short, even though free from sulphur, or other substances which might themselves produce the effect. It suffices that so small a quantity as 0.25 per cent. of manganese shall enter into the composition of the metal to render it forgeable ; but, if the manganese be removed by the addition of oxide of iron, or by exposing the metal to the air, red-shortness will again ensue, only to be removed by a further addition of manganese, thus indicating that the *presence* of manganese in iron which has undergone fusion is absolutely essential to the prevention of red-shortness. The reason of this has yet to be made out.

In practice, the amount of manganese which has to be added to render the metal forgeable is found to depend on the amount of sulphur present, since the ores of iron, with few exceptions, contain sulphur in sufficient quantity to affect the quality of the metal produced from them, if not neutralised by a sufficient quantity of manganese. It is not clear in what way manganese acts in destroying the deleterious effect of sulphur, for it has been proved by analysis that, at any rate in the generality of cases, it does not carry the sulphur from the metal into the slag. It should be unnecessary to remark that phosphorus is not removed at all in this process, since the furnace has a silicious lining.

By the Siemens-Martin and the Siemens processes metal of any desired composition can be produced ; in this country the Siemens process is, however, almost entirely confined to the manufacture of soft metal, very low in carbon (0.13 to 0.24), for boiler-plates, ship-plates, and angle iron ; almost all the rail metal being manufactured by the Bessemer process, hereafter to be described. In France a

good deal of rail metal is made by the pig and scrap process. The make of a furnace is about the same, whichever method is worked—70 to 90 tons per week for a 5-ton furnace ; but rather more fuel is required for the ore process, owing to there being a greater quantity of slag, which, being a bad conductor of heat, causes the consumption of a greater amount of fuel to raise the metal to the proper temperature. The consumption of fuel in the ore process may be taken at about 15 cwt. per ton of metal made. In consequence of the greater intensity of the heat required in the ore process, the wear of the furnace is greater, more particularly of the roof, in the destruction of which the slag splashed on to it also plays a not unimportant part. A roof made of good Dinas brick will, however, if properly used, last from 150 to 200 charges. The tendency has been to increase the size of the furnaces. At first they were commonly made for 5-ton charges, now furnaces which will take 10 tons are common, and some 15 and 20 ton furnaces have recently been erected. So long as everything is working properly, there would appear to be rather an advantage than otherwise in using these large furnaces, but should anything go wrong, the inconvenience and loss would be very considerable. The limit to the advantage to be obtained by increase of size has yet to be determined.

Notwithstanding the success which has attended the introduction of the process bearing his name, Dr. Siemens has continued to strive after even a more perfect method of producing malleable metal from iron ore. His object has been to do away with the intervention of the blast-furnace, substituting some method by means of which ordinary ores, containing phosphorus, may be directly converted into malleable iron, free from that objectionable substance ; the fusion and carburisation of the metal to be effected in the regenerative gas furnace in the ordinary way. We cannot do better than quote here Dr. Siemens'

own words. He says : 'In my early experiments in this direction I followed the lead of Chenot and others, in producing what is called spongy iron, or iron deprived of its oxygen by heating it to redness in combination with carbonaceous material. I soon convinced myself, however, that no practical results could be obtained by this means, inasmuch as the spongy iron contains, bound up with it, the gangue of the ore, which can with difficulty be separated

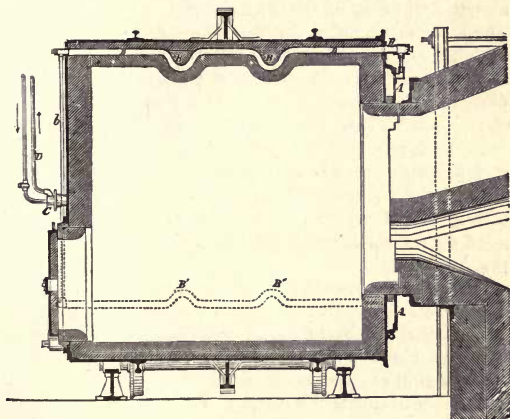


FIG. 48.

from the metal, and afterwards encumbers the melting-furnace with excessive slag. All the hurtful impurities contained in the ore, such as sulphur, phosphorus, arsenic, &c. remain, moreover, in the spongy iron ; and, as regards sulphur, its quantity is much increased on account of a powerful absorbing action, exercised by the spongy iron upon the sulphurous acid contained in the flame of the furnace. It was necessary, therefore, to devise some plan by which the metallic iron could be simultaneously separated

both from the ore and its impurities. This object I have succeeded in accomplishing by means of a rotating furnace. The furnace, which is represented in Figs. 48 and 49, consists of a gas producer, the air regenerator, a reversing valve, and the revolving drum. No gas regenerators are employed in this furnace, but the gas passes from the producers through an oblong channel continuously into the revolving chamber, where it is brought into contact with the heated current of air passing in from one or the other of the air regenerators. The flame thus produced rushes forward into the heating-chamber, and after heating the material therein, passes back again towards the inlet side, whence the products of combustion pass through the second air regenerator and the reversing valve into the chimney stack. By this arrangement the front of the rotatory furnace is left free for access, and is provided with a charging and discharging door, placed eccentrically, for the convenience of withdrawing masses or balls of iron from the furnace on a level with the lining, when the furnace is stopped with the aperture in its lowest position.'

In order to prevent the semi-fluid or partially molten mass from sliding along the lining, instead of being rolled over on itself, so as to expose continually a fresh surface, and with the view to protect the throat of the furnace, which is most exposed to the destructive action of the heat, and further to ensure the division of the puddled mass into balls of convenient size for handling, the following arrangements are made. At the back of the rotating vessel is an annular casing (A) surrounding the throat, from which two or more pipes (B) are led along and within the refractory lining to a trunnion joint at the centre of the front of the vessel. In this trunnion joint is provided a facing (C) like that of a rotating slide-valve, with ports connected with the circulation pipes, and also with a supply and a discharge pipe (D), connected to the trunnion. As the vessel revolves, water flows by the trunnion ports along some of the pipes to the

annular casing at the throat of the vessel, and back thence to the discharge, the circulation keeping the pipes and the throat comparatively cool. The low temperature of the pipes produces a corresponding cooling of the lining in their vicinity, causing some of the semi-molten material in the furnace to solidify, forming longitudinal ridges, which, as the vessel rotates, make the pasty mass to roll over on itself, thereby facilitating and promoting the reducing and puddling action. In order to bring about the subdivision

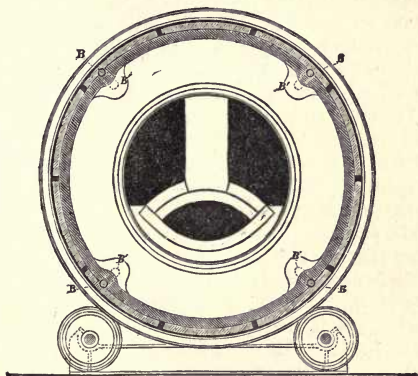


FIG. 49.

of the mass so as to form balls of convenient size, the pipes are constructed with several bends inwards (B'). The lining of the furnace is made of highly aluminous or bauxite bricks, covered on the inside with a certain thickness of iron oxide, produced by melting hammer scale and rich ore in the furnace, which set while the furnace is kept slowly rotating. The rotation of the furnace is effected by means of a small Brotherhood or other engine, and suitable gearing.

The *modus operandi* is as follows : The furnace, being

already lined and heated, a batch of ore, mixed with fluxing and reducing materials, in a proportion depending upon the chemical constitution of the ore employed, is charged from an elevated platform in front, the rotator being stopped for this purpose with the charging orifice in its upper position. From 30 to 40 cwt. of batch is thus charged, upon which the door is closed, and the furnace chamber is made to rotate at the very slow rate of six or eight revolutions per hour. A high temperature is produced within the chamber by the combination of the gas with the highly heated air from the regenerator, causing the mass rapidly to become hot, whilst slowly rotating so as to present continually new surfaces to the heat. No chemical action takes place under these circumstances until the temperature of the mass is raised to a full red heat, when reaction between the carbonaceous matter and the ore will take place, giving rise to the development of carbonic oxide, which, meeting the heated air proceeding from the regenerators, is burned, and thus adds to the heating action of the flame. When this reaction has fully set in, the supply of producer gas may be almost entirely stopped, and thus no sulphurous gas is admitted into the furnace during this critical interval. The heat now rises rapidly, and fusion of the earthy constituents of the ore occurs simultaneously with a continuance of the reducing action. In the course of an hour and a half after starting, the charge consists of metallic iron in a more or less agglomerated condition, found on analysis to be almost chemically pure, and of a liquid mass of cinder containing the earthy constituents of the ore, and other foreign matter. The rotation of the furnace is now stopped with the tapping-hole in its lowest position, and the bulk of the cinder is discharged; the tapping-hole is thereupon closed again, and the furnace made to rotate somewhat more rapidly, with a view of facilitating the agglomeration of the metallic iron; by the timely introduction of a rabble, the agglomeration of the mass can be so regulated as to induce the

formation of two or three balls of convenient size for handling. The balls being formed, the furnace is stopped with the large door in its lowest position, which, upon being removed, admits of the charge being withdrawn. This is effected by the introduction of tongs supported by pulleys running upon overhead rails, for transferring the balls in rapid succession from the furnace to a squeezer (which has for its purpose to expel adhering cinder), and from the squeezer the balls are carried to the bath of a steel-melting furnace, where they are fused with a little pig to increase the fluidity.

The great purity of the metal thus reduced from the ore, and the rapid and comparatively inexpensive nature of the reducing process, appear to be conditions favourable to the production of steel of high quality by this method at reasonable cost.

According to the late Mr. A. L. Holley, the eminent American metallurgist, the Siemens rotators at Pittsburg were producing remarkably pure iron *at the cost of pig*.

In 1876 a patent was taken out by M. Pernot, in which it was proposed to produce steel in an open-hearth regenerative furnace with a revolving bed, inclined at an angle of 5° or 6° to the vertical (Fig. 50). Pig-iron, previously heated to redness, is placed on the bed of the furnace, and covered with scrap. The bed of the furnace is then made to revolve slowly, the pig gradually melts, and the scrap is alternately exposed by the motion of the furnace to the strong heat of the flame, and then dipped under the molten pig-iron. In this way the fusion is very rapid comparatively, the whole mass becoming fluid in about two hours. The process is completed in the ordinary way.

When repairs are necessary, the bed on its carriage is drawn out.

In practice it is found that these furnaces require very frequent repairing. With the view to make this easier, M. Pernot has arranged a movable roof, which has, besides,

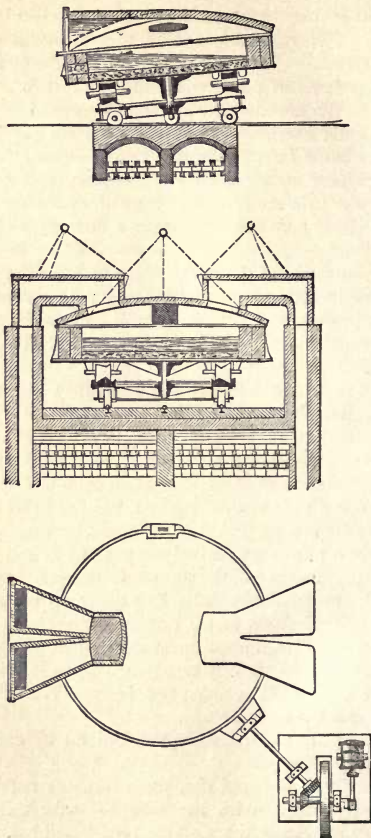


FIG. 52

the additional advantage of reducing somewhat the strain on the structure occasioned by such great variations in temperature. M. Pernot has recently taken out a further patent for an arrangement of his furnace by means of which he can employ gas under pressure. He states that he has obtained by this means results which have never been equalled before, viz. five charges from a 7-ton furnace in twenty-four hours.

Leaving out of consideration these more recent forms of Pernot furnace, of the efficiency of which we have not yet had sufficient opportunity to judge, and confining our attention to the original arrangement of the furnace, it is important to note that, notwithstanding all that is claimed for it, it has not succeeded in establishing itself in this country. An enterprising firm in Scotland, who are working the Siemens process very largely, put up a Pernot furnace. They state that they gave it a fair trial, and were not satisfied until they had obtained as good a make from it as was said to be obtained in France. Yet they abandoned its use, giving as their reason that in the long run they found it had no advantage over the ordinary Siemens furnace in point of make, and the trouble in working it and the expense for repairs were greater. It is only fair, however, to point out that, in the case instanced, the furnace was employed for the ore process, and for soft metal requiring a high temperature to keep it molten, whereas in France, and also in America, where it is stated to have been successfully worked, it is used for the pig and scrap method, for rail metal containing about 0.4 of carbon. Herein lies an important difference, which should not be overlooked. There appears to be no reason to dispute that in the Pernot furnace a charge can be worked off more rapidly than in the ordinary Siemens furnace, and that consequently, in addition to saving in time and labour, there is an appreciable saving in fuel; but it must be remembered that, in working at such extremely high temperatures, there are serious mechanical difficulties opposed

to even the simplest construction of furnace holding together, and that these difficulties are enormously increased in dealing with a furnace of more complicated structure. Thus it may well happen that the time and interest on capital lost whilst the furnace is undergoing repairs, and the additional expense for repairs, may be greater than can be balanced by the advantages of the system. The wear and tear in the ore process, as already pointed out, is greater on account of the higher temperature and larger quantity of slag than in the Siemens-Martin process, and this difference may be about equal to or greater than the margin of advantage which there may be in favour of the Pernot furnace when working on pig and scrap without ore. It will have been observed that the dominant idea in treating pig-iron for the production of malleable iron or steel has always been to refine the metal by the action of atmospheric air, and this was effected by causing a current of air to impinge upon the surface of the metal, by means either of blowing apparatus or the drawing action of a chimney-stack. What more natural than that it should occur to some one to refine iron by blowing air *into* it, instead of merely on to its surface? We find that this idea did actually occur to several persons in the year 1855, shortly before the introduction of the regenerative system, which we have seen has borne such important fruits. Not less important were the results destined to follow the, at first sight, not very important modification in the method of applying air to the refining of iron. The now world-famed Bessemer process resulted from the attempt to carry out this idea. Until it demonstrated itself, it was not for a moment suspected by anyone—even by Bessemer himself—what an enormously high temperature could be produced by simply blowing air into molten pig-iron. He started with the idea of refining pig as a preliminary operation to puddling, and had no idea that the temperature produced would be sufficient to maintain in a molten condition the metal after it had been completely decarburised;

a state of things, be it remarked, never before attained to. The discovery made by Bessemer was worked out by him in a most masterly manner, showing his great mechanical genius. The machinery invented by him for the carrying out of the process has remained in use ever since, only one or two appreciable alterations having been made, although hundreds of able engineers have since had to do with the process in various parts of the world, millions of tons of forgeable metal having been made by it. Fixed *converters* were at first employed by Bessemer, but he soon superseded them in this country by converters movable on trunnions by hydraulic or steam power (Figs. 51, 52), which are now universally adopted. The converter is a pear-shaped vessel, varying in size, but commonly about 8 ft. diameter \times 15 ft. high, constructed of boiler-plate, lined to a thickness of about 1 foot with ground ganister. In the bottom are arranged from 7 to 12 twyers, consisting of fire-clay cylinders, pierced with from 7 to 12 holes about $\frac{3}{8}$ inch in diameter. The twyer-box is connected by a curved pipe with one arm of the trunnion, which is hollow, and in its turn is connected with the blast main, so that air at a pressure of from 15 to 25 lbs. can at will be forced through the twyers into the interior of the converting vessel. The other arm of the trunnion is solid, and has connected to it gearing by means of which the converter can be turned over (Fig. 52), so as to enable the contents to be poured out or a fresh charge run in. In order to avoid delay, in the most recent practice the vessel is divided into four parts besides a loose bottom; spare pieces corresponding to these, with the exception of the trunnion section, are kept in stock, ready lined and dried, so that they can be substituted for those requiring repair in a very short space of time, leaving only the trunnion section to be repaired in position. Before charging the metal in, the converter is heated to redness by coke being burnt in it by a gentle blast of air.

The pig-iron to be converted may be taken straight from

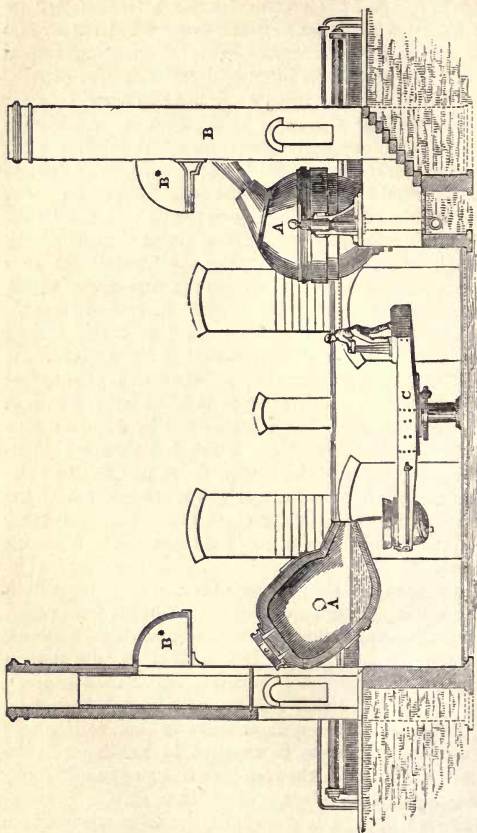


FIG. 51.—Bessemer's process. A, Converting vessel. B*, Hood for carrying the gases into the chimney B. C, Crane for swinging the ladle under the converter. (Shallower pits than that shown in the figure are now used.)

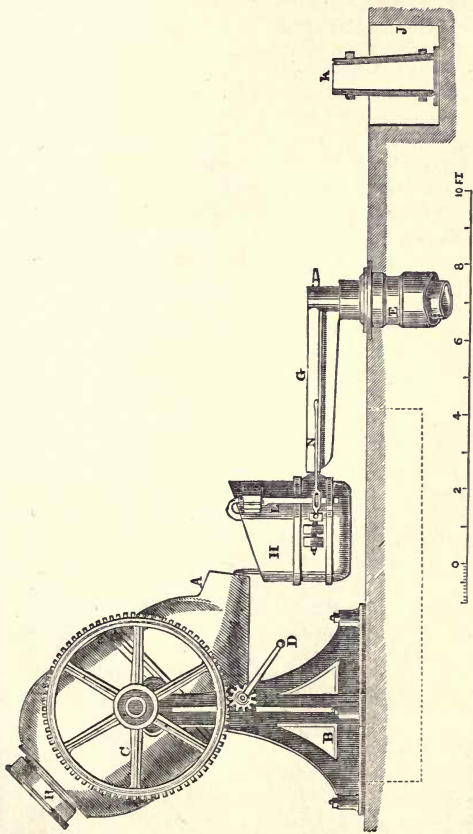


FIG. 52.—Operation of transferring Bessemer steel from the converter to the ladle. A, The converter. B, Standards supporting the trunnions or axes of the converter. C, Spur-wheel for tipping the converter by means of the pinion moved by the handle D. E E, Hydraulic crane for swinging the casting-ladle H. J, Casting-pit. K, One of the ingot-moulds. L, Rod terminating in a conical fire-clay plug, which closes the opening in the bottom of the casting-ladle, and may be raised by the handle N to permit the flow of the metal into the moulds.

the blast-furnaces to the converter in large ladles mounted on a carriage drawn by an engine, the ladle being constructed so that it can be lifted off the carriage by a crane, and its contents tipped into the converter; or pigs of iron may be melted in a cupola near the converter, and the metal run thence along a channel into the converter, or a ladle may be employed to transfer it from the cupola to the converter. Obviously there is a considerable amount of fuel and labour saved by transferring the metal direct from the blast-furnace to the converter, but it can only be conveniently done in practice when a supply can be obtained from several blast-furnaces. The metal from a single blast-furnace is liable to vary considerably in composition from day to day; by mixing the metal from several blast-furnaces in suitable proportions a metal of the required composition can always be obtained, and no difficulty is experienced in the converting process, such as would arise were the character of the metal constantly varying.

When the molten pig-iron is received at the converter a little is poured into a small mould, cooled quickly in water, and fractured. By the character of the fracture the quality of the pig and the treatment it will require can be sufficiently nearly judged. The metal having been run into the converter, the blast is put on, and the vessel turned into the upright position. The pressure of the blast must be sufficient not only to sustain the column of metal, but to force its way through it. The pig iron, it will be remembered, contains, besides iron, the following substances in more or less quantity: silicon, manganese, carbon, phosphorus, and sulphur, all of which, when heated, are capable of combining with oxygen. At the commencement of the 'blow' the action of the blast is confined mainly to the silicon and manganese, only a little carbon and iron being burnt; consequently there is but little flame, the products of the combustion of silicon and manganese, viz. silica and oxide of manganese, being solids, which remain in the converter as slag, whilst the carbon issues from the

metal already oxidised to the maximum. Soon, however, the silicon and manganese are burnt out, and the carbon is vigorously attacked. The roaring of the blast, the rush of white-hot flame from the mouth of the converter, occasionally accompanied by slag, which is projected far into the air and falls in a golden shower, varied with scintillating particles of burning iron, are very impressive to witness, especially at night, when the dark shadows lend additional grandeur to the scene. This period, during which the carbon is being burnt out, is known as 'the boil;' the carbon is now only oxidised to carbonic oxide, the temperature being so high that carbonic acid would be decomposed, hence the flame at the mouth of the converter. The point at which the carbon has all been removed is recognised by the dropping of the flame. Phosphorus is not removed in this process to any appreciable extent, owing to the lining of the converter being silicious. Pig-iron free from phosphorus has therefore to be employed. The action of silica in preventing the removal of phosphorus has already been explained. Sulphur is very difficult of removal, and should therefore be present only in very small quantity, as a great loss of iron would attend its removal, and, if allowed to remain, the character of the metal would be deteriorated. It follows that, directly the flame drops the converter must be turned down and the blowing stopped, otherwise the iron would be rapidly oxidised, and issue from the mouth of the converter as a thick brown smoke. It was soon discovered that workable metal could not be made by this process without the aid of manganese, no matter whether some carbon were left in or it were all removed. The credit of suggesting the introduction of the manganese in the form of spiegel-eisen is due to Robert Mushet. We have seen how, afterwards, spiegel was employed with a similar object in the Siemens process. After the converter has been turned down, the carbon having been reduced to about 0.10 per cent. or less, 5 to 10 per cent. of melted spiegel is run in, causing a burst

of flame, due to the action of the carbon in the spiegel on the oxygen in the blown metal. In some works the converter is turned up again, and the blowing continued for a few seconds ; more commonly the metal is simply poured out into the ladle, whence it is run into the ingot moulds, it being considered that in this way the manganese becomes sufficiently and uniformly distributed throughout the iron. That a perfectly homogeneous metal is not, however, produced in this way has been proved beyond a doubt. It so happens that the want of uniformity of composition is rarely so great as to destroy the value of the metal for rails, and such like purposes, although the tests are severe ; but for boiler-plates and cutlery steel the case is undoubtedly quite otherwise. Bessemer foresaw this so long ago as 1863, and provided a remedy, which, however, with one sole exception has never been applied, and in that case only within the last four years. The remedy is to agitate the metal by a mechanical stirrer after the addition of the spiegel. The expedient is simple and inexpensive, and has been found to be very effective. Fig. 53 illustrates the contrivance, and speaks for itself as to the arrangement of the parts. The agitator is constructed in iron or steel, and coated with loam or ganister. The ladle containing the metal, which has just been poured out from the converter, is carried under the agitator by an hydraulic crane, and then raised so that the agitator is immersed, as shown in the figure. The stirrer is then revolved at about 100 times a minute. Occluded gases are found to be expelled in great quantity, the metal finally becoming quite tranquil. The whole operation only requires two or three minutes, and at its completion the metal can be run into the moulds, where it will lie quietly, and solidify into sound ingots. On the other hand, when mechanical stirring has not been resorted to, the metal, after it has been teemed into the moulds, tends to rise and form unsound ingots. In order to prevent this, as far as possible, sand is thrown on the top of the metal directly the mould is full,

and a piece of sheet iron is placed on the top, being held down by a bar kept in position by hooks on the side of the mould, wedges being driven under the bar. This device is, as might be expected, far from completely successful in preventing honeycombing, though the pressure induced does to some extent prevent the formation of bubbles. The gas,

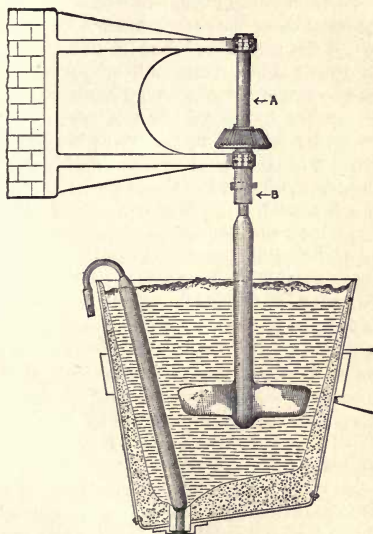


FIG. 53.

no matter whether it remains entrapped in the metal or is allowed to escape during solidification, must do mischief. It will thus be seen that there are two sources of evil : the one want of homogeneity, the other porosity. These are both due to a great extent to one and the same cause—want of

proper mixing. If the spiegel have not been properly mixed in with the blown metal by the time it is in the moulds, the product will consist of a mixture of metal still containing oxygen, and of carburised metal ; but during the cooling of the mass, and by reason of it, there will be a circulation going on, constantly bringing some of the metal still containing oxygen in contact with that which has been carburised, the result being that gas continues to be evolved up to the time that the metal sets. Besides carbonic oxide, steel tends to retain a large quantity of hydrogen, produced from the decomposition of aqueous vapour in the air ; this also is probably expelled by the agitator. It has been attempted to get over the difficulty in respect to honeycombing by subjecting the metal to great pressure during solidification in the mould. This plan has been very successfully carried out, but it has not hitherto been found applicable in practice where a large number of ingots are being rapidly cast, as in the manufacture of rail-metal. In Whitworth's method the metal is poured into an annular steel mould of great strength, into which fits an annular ram connected with the piston of an hydraulic press ; when the mould is full, the ram is brought down upon the metal with a pressure which is increased up to several tons per square inch, at which it is maintained until the whole of the metal in the mould has solidified. The method introduced by M. Considère in France, and about the same time by Mr. H. R. Jones in America, consists in applying the pressure by means of steam conducted from a high-pressure boiler by a flexible pipe to the ingot mould, which is supplied with a movable lid, capable of being tightly keyed down. Of the two systems the latter would appear to be easier of general application, and more likely to insure the continuance of the pressure on the metal in the centre of the ingot up to the moment of solidification ; a point of considerable importance in preventing the formation of a central cavity. Steam at a pressure of about 100 lbs. on the square inch is said to be

sufficient for rail-steel, a somewhat higher pressure being required for mild steel.

When the ingot has sufficiently cooled, the mould is lifted off by a crane ; the ingot is then placed on a trolley and carried to a heating furnace, where it is allowed to attain the necessary temperature, when it is withdrawn and put through the blooming-rolls, after which it is transferred to a reheating furnace, and finally it is rolled into the required shape.

It will be of great interest and practical importance to trace to its source the very high temperature which we have seen is produced in this process. The substances which are burnt out are silicon, manganese, carbon, and a little iron. The last named may be left out of consideration, as it is not essential to the process, care being taken to avoid, as far as possible, the combustion of the iron. Of the three remaining substances, silicon is that which contributes the greatest amount of heat, manganese coming next, and carbon last. The carbon is seriously handicapped, owing to its product of combustion being a gas, causing a very considerable absorption of heat-energy, whilst in the case of the combustion of silicon and of manganese the whole of the heat of combustion is available for raising the temperature of the charge. Were silicon and manganese absent from the pig-iron, the temperature produced by the combustion of the carbon alone would not be sufficient to render the charge properly fluid. It will be shown presently that a too high temperature, as well as a too low temperature, may be obtained under certain conditions. Apart from the necessity of making the metal sufficiently fluid to pour clean from the converter, a moderately high temperature produces a metal of a homogeneous character and close texture, not otherwise obtained. There are three considerations by which the temperature is mainly regulated : the quantity of metal, its composition, and the rapidity of the blow. The quantity of the metal used in each blow and the rapidity of

the blow are well within control ; the composition of pig-iron is very much less so. It thus happens that, in England, it is the constant aim of the Bessemer-steel maker to keep the percentage of the silicon in the pig as low as he can, whilst in Sweden and parts of Germany just the reverse is the case. This great difference depends solely on the fact that in the former case coke is used in the blast-furnace, and ores containing but little manganese, whilst in the latter charcoal is used, and manganese is commonly present. The coke contains always more or less sulphur, not uncommonly 1 per cent. It has already been pointed out that, in order to prevent pig-iron from taking up sulphur, it must be made very grey ; but the greyer the iron the more silicon will it contain. Thus the Bessemer-pig maker using coke, so to speak, sits between two stools. On the one hand is sulphur, on the other silicon. If the grade of the iron be allowed to go too low, it will contain too much sulphur, which not being removed, except to a very small extent, in the converter, will cause the steel to be red-short. If the grade of the pig be too high, there will be so much silicon in it that it would cause the blow to be too hot, to avoid which a quantity of cold scrap has to be thrown into the converter. There is no objection to the presence of silicon, so long as it is not in so great a quantity as to require more scrap than can be conveniently and economically supplied by the works. The difficulty can be conveniently got over in large works by mixing different grades of pig whilst still molten. If the metal be blown too hot, owing to an excess of silicon, too much of that element will remain in the finished product, causing its quality to be deteriorated ; silicon in quantities exceeding about 0.50 per cent. tending to produce both red- and cold-shortness. The exact amount of silicon, however, which is permissible will depend on the relative proportions of carbon, manganese, phosphorus, and sulphur also present, and on the purpose for which the steel or ingot-iron is to be employed.

If charcoal be used as the fuel in the blast-furnace, with ore fairly free from sulphur, and some manganese be present, the sulphur difficulty is got rid of; but when charcoal is used the conditions in the blast-furnace are unfavourable to obtaining a pig sufficiently rich in silicon. The difficulty could, probably, be most satisfactorily solved by mixing coke-pig, high in silicon and free from sulphur, with charcoal-pig. Charcoal-pig must, however, in the natural order of things, before many years have passed over our heads, become a scarcity in Europe and America.

Pig-iron, containing from 2 to 4 per cent. of manganese, can be worked by what is known as the direct process, when it is only desired to reduce the carbon to 0·3 or 0·4 per cent., as for rails; *i.e.* the blow is continued until the carbon is reduced to the requisite amount, the manganese still remaining in (0·1 to 0·3) being sufficient to render the metal workable. This method is in use in Sweden, and parts of Germany, where suitable pig is available. It would not pay to make pig specially for the purpose by introducing manganese into the blast-furnace. The exact moment at which the blow should be stopped can only be judged of by the spectroscope, and by observing the colour of the slag and the forgeability of globules of metal rapidly withdrawn from the converter on the end of an iron bar.

The average time required to blow a 7-ton charge in England for rails is about twenty minutes.

Notwithstanding the fact that phosphorus cannot be eliminated in the ordinary Bessemer converter, enormous quantities of metal have been made by this process. Means have happily recently been devised by which this bugbear of Bessemer-steel makers has been overcome. The process is known as the Thomas-Gilchrist or 'basic' process.

In the ordinary Bessemer converter we have seen that the lining is formed of ganister, a silicious material. Now, it has been shown that silica has a greater affinity for oxide of iron than phosphoric acid has; consequently, so long as

free silica is present, phosphoric acid cannot remain in combination with oxide of iron; whilst, then, the lining of the converter was of silica, it is sufficiently obvious that phosphoric acid could never be eliminated.

You will at once say, why line the converter with this objectionable substance? The answer is easy: no substitute was known, and the reason why phosphorus was not removed was not generally understood.

This was the state of things when Messrs. Thomas and Gilchrist commenced their experiments. The object they had in view was to substitute for the ganister a basic material such as lime. The difficulty they had to contend with was to obtain a lining which would hold together. After many failures and much patient labour, a material has been found which fulfils the necessary conditions. This material is magnesian limestone. By grinding it and mixing it with pitch, as patented by Mr. Riley, bricks can be formed which, after burning, are very refractory. In lining the converter it was impossible to cement the bricks satisfactorily together: they generally get a good deal curved in baking, and fit badly together, and the cementing material is easily washed out by the molten metal. In order to get over this difficulty, the converters are now lined by placing a hollow tapered iron core in the centre, at the bottom of which a coke fire is lit, a liquid mixture, or 'slurry,' of tar and previously well-burnt limestone is run into the space between the core and the shell of the converter, which has had a layer of bricks placed flatwise against it. The lining consolidates with the heat, so to speak, into one solid brick. This method has proved to be very successful.

From an enemy, by judicious treatment, we may be said to have converted phosphorus into a friend. In the 'acid' process it is essential that about 2 per cent. of silicon should be present, for it is, to a great extent, due to the presence of silicon that the requisite high temperature can be obtained. In the basic process the less silicon there is the better,

because it destroys the lining by fluxing it away. Here it is that the phosphorus befriends us, for it, too is capable of producing a high temperature by combining with oxygen; and that being the case, it becomes possible to work with about half the silicon necessary in the acid process, which practically means that we may employ a much lower grade of iron, for the lower the grade of iron the smaller will be the amount of silicon in it.

The phosphorus is removed by continuing the blow after the flame has dropped, *i.e.* after the carbon has been all burnt out. This is called the 'after-blow.' The time the after-blow should last can only be regulated by experience gained by direct experiment with the particular class of iron being treated. The maximum time which the after-blow should last being known, the converter is turned down before that time has expired, and a small sample taken, which is hammered out and fractured. A little practice soon enables the workman to decide with great accuracy from these rapidly performed tests how many seconds longer the blowing should be continued in order to effect the entire removal of the phosphorus.

So appreciated has this hitherto despised substance become, that it is actually the practice to put back into the blast-furnace a great part of the slag from the converter, in order to increase the amount of phosphorus in the pig-iron subsequently to be converted into steel.

There is, however, a limit to the lowness of grade of iron which can be used, for although the silicon decreases, the sulphur increases, and only about half the sulphur present in the pig-iron can be removed in the converter. One-tenth per cent. of sulphur suffices to prevent Bessemer or Siemens steel from rolling in a sound condition. As has already been pointed out, the way to counteract the influence of sulphur is to employ manganese in sufficient quantity, but this is not without a serious drawback, for manganese is expensive.

In working out this process much difficulty was at first

experienced owing to the mouth of the converter getting gobbled up, that is to say, stopped by projected slag. The basic slag, consisting as it does principally of lime, is very pasty. This inconvenience has been successfully got over by employing converters of the form shown in Fig. 54. It was predicted by many that the slag and metal would be thrown out of the mouth of this form of converter, but that has not been the case, and it is not improbable that eventually this shaped vessel will be universally adopted for both

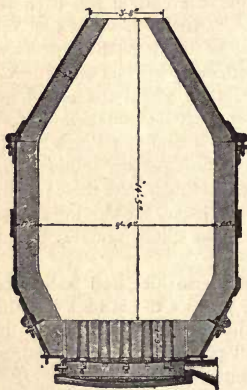


FIG. 54.

the acid and the basic processes. Such is already the case at Messrs. Bolckow, Vaughan & Co.'s works, where, under the intelligent and persevering guidance of Mr. Windsor Richards, the basic process has first been made a commercial success in this country.

On the Continent this process is being largely worked, and with very great success. Owing to the ease with which the phosphorus can be completely eliminated, and the small amount of silicon retained in the finished product, the

quantity originally present in the pig being much less than in the acid process, a metal of remarkable malleability and ductility can be obtained, possessing, according to Kupelweiser, a conductivity for electricity only rivalled by pure iron specially prepared in a laboratory.

By a simple device of the well-known American metallurgist, A. L. Holley, lately deceased, the make of a basic plant has been greatly increased. The arrangement con-

sists in making the converter quite independent of the trunnion, so that, when repairs are required, the converter can be rapidly removed and another put in its place. A car is run under the shell, and raised by an hydraulic lift until it takes the weight of the latter off the trunnion ring, which is then disconnected by knocking out the cotters, and lifted above the shell. The shell is lowered and taken away to be repaired, another being raised under the trunnion ring, which is then lowered into place and cottered, the blast joint at the same time being screwed up.

In France, a basic lining has been employed in the Siemens-Martin process, with reported success.

A few years since a furnace was patented by M. Ponsard which aims at combining the advantages of the Bessemer and open-hearth processes. It is a revolving reverberatory furnace supplied with twyers. By giving the furnace a half revolution on its oblique axis, the twyers may be brought either beneath or above the surface of the bath of metal. By these means the metal can be rapidly decarburised nearly entirely, as in the Bessemer converter, and then, by removing the twyers from beneath the metal, the final adjustment of the carbon can be made, as in the Siemens process. The chief difficulty experienced in working out this idea to a practical success appears to be the rapid destruction of the twyers. This obstacle is certainly a very great one, and may prove insurmountable. A further discouragement to the attempt to work in this direction is to be found in the success of the basic-Bessemer process, by means of which a metal, the character of which is well within control, can be produced.

In consequence of the extremely high temperature which we can command, either in the Bessemer or open-hearth processes, it is possible to obtain, in a molten state, metal practically free from carbon, or containing carbon to any required amount. It is sufficiently obvious that, having regard to the original and commonly understood meaning of the word

steel, some other name should, strictly speaking, be applied to all metal manufactured by these processes which cannot be hardened and tempered. In practice, however, there are many obstacles in the way of this being done, and it has become customary in this country to designate by the term steel all the metal which has been produced by the Bessemer or open-hearth processes.

It thus has resulted that we speak of steel ships, steel boilers, and steel rails. The metal of which ship- and boiler-plates are made contains about $\frac{1\frac{3}{10}}{100}$ ths to $\frac{2\frac{4}{10}}{100}$ ths per cent. of carbon, while rails usually have about $\frac{4}{100}$ ths. The first could not be appreciably hardened, and the second is considerably below what would formerly have been considered steel.

Although, then, metal possessing the true characteristics of steel can be made by these processes, yet that which is ordinarily made is not steel, but a metal called into existence by our recently acquired power of obtaining an extremely high temperature.

This new metal, as we may fairly call it, has properties far excelling those of wrought iron, and it has only been a question of time to make this universally felt.

At the present moment new iron rails are things of the past, and wooden sleepers have begun to follow in their wake, it having become apparent to all that our new metal will be an economical substitute. So with ships: the wooden walls of old England are no more. The so-called steel is not only supplanting wrought iron where it was used, but wood also.

There is now but one sound reason why steel should not universally replace iron with advantage, and that is, that in some cases it is cheaper to employ iron. Statistics show us that the enormous quantities of steel now manufactured have but little, if at all, affected the production of wrought iron. It is, however, but a question of time when the day will come, and every day brings us nearer to it, when steel will be manufactured as cheaply as iron. Then will wrought iron be a thing of the past amongst the great civilised nations.

THE PHYSICAL PROPERTIES OF COMBINATIONS OF IRON
WITH OTHER ELEMENTS.

There are great practical difficulties in the way of studying the compounds of iron with each individual element. The knowledge which we at present possess is entirely derived from the observation of the effect of varying each element, when several are present.

By far the most important of all the combinations of iron with other elements, which we have to consider, is that with carbon in varying proportions.

The effect of the presence of *carbon* in iron is, up to a certain point, to increase the tenacity and decrease the ductility, and to cause the metal, when heated and then cooled more or less suddenly, to harden, the hardening being in direct proportion to the amount of carbon present and to the rate of cooling. The presence of some other elements, such as silicon, manganese, and phosphorus, increases the hardening effect of a given amount of carbon. Metal containing less than 0.75 per cent. of carbon is not usually employed in cases where it is intended to harden, the amount of hardening producible with less than that percentage being usually insufficient. On the other hand, metal containing about 1.5 per cent. of carbon represents the maximum which can be safely worked, even then very great care being required to prevent its becoming overheated and burnt. Steel of this degree of carburisation is used for razors and tools for turning chilled rolls. In hardening it, it should not be heated above a dull cherry-red. Steel with about one per cent. of carbon is extremely useful, as it can be welded without much difficulty, and the unhardened portion of a tool made from it is sufficiently tough to withstand the blow of a hammer without snapping. Cold chisels and hot setts are made of this class of steel. Of the intermediate tempers between $1\frac{1}{2}$ and 1 per cent. may be mentioned the following:— $1\frac{3}{8}$ per cent., temper for saw files; $1\frac{1}{4}$ per cent., tool temper, e.g. drills, turning

and planing machine tools ; it requires great skill to weld steel of this temper. $1\frac{1}{8}$ per cent. is spindle temper, a very good temper for the largest size of turning tools, hot setts, millpicks, circular cutters, &c. The chief uses of the tempers below 1 per cent. are for tools which have to stand very heavy blows, or great and sustained pressure on the unhardened portion, as in the case of cold setts, dies, shears, &c.

The temper of steel depends not only on the degree of carburisation, but also on the rate of cooling. The usual method of procedure is to harden the steel to the maximum, and then *temper* it. The hardening is effected by plunging into water, or a solution of some salt, the piece of steel which has been carefully heated to the temperature suited to its degree of carburisation. The result of this operation is to make the metal hard, but inelastic. In order to obtain the latter much-prized property of steel a distinct operation called *tempering* is resorted to. This consists in reheating the steel to a certain temperature, which is regulated by the temper required, and then allowing it to cool slowly. When reheated, as the temperature rises the surface of the steel passes gradually through a series of beautiful colours—due to films of oxide of iron. These colours indicate to the experienced eye of the workman the temperature at which the steel has arrived, and therefore the point at which to stop the reheating. The approximate temperatures corresponding to the various tints are shown in the following table :

Temp. F.	Colour	Temper
430	Very faint yellow.	Lancets.
450	Pale straw.	Razors and surgical instruments.
470	Full yellow.	Penknives.
490	Brown.	Scissors and cold chisels.
510	{ Brown, with purple } spots.	Axes, plane irons, pocket-knives.
530	Purple.	Table-knives, large shears
550	Bright blue.	Swords, watch-springs.
560	Full blue.	Fine saws, augers.
600	Dark blue.	Hand and pit saws.

During the hardening process the tool should not be removed from the water until it is quite cold, otherwise it is liable to crack. When it is intended to harden the edge of a tool only, the metal should be moved slightly up and down in the water, so as to avoid an abrupt transition from the hardened to the unhardened part, which would be likely to cause it to crack. Over-heating at some time or other is a common cause of steel cracking during hardening. It is also sometimes due to 'over-melting.' The same result may be traced to the presence of too much manganese. Steel containing phosphorus will behave in a similar manner.

By hardening in oil, which, owing to its low specific heat, abstracts the heat from the metal very slowly, the tempering stage is rendered unnecessary for some purposes.

By hardening, steel is expanded from its original size when cold, the expansion being greater the higher the temperature to which the steel has been raised and the more quickly the cooling is effected. Notwithstanding this, a given piece of steel may be smaller after it has been hardened than it was before. This apparent anomaly is not difficult of explanation. In pieces of steel above a certain size the hardening does not extend right through to the centre. The surface, when it is suddenly cooled, contracts to a certain extent, and exerts a considerable compressive force on the metal in the interior, which, as it slowly cools, is forced to occupy a smaller volume than it did originally; whilst the hardened portion, which is in a state of tension owing to its having been cooled suddenly, occupies a greater. If then the contraction of the interior be greater than the expansion of the exterior, the piece of steel as a whole will be smaller after hardening than it was before, and *vice versa*. The whole question turns on the relation of the volume of the hardened portion to that which has been only partially hardened.

Silicon.—This substance produces both red-shortness and cold-shortness. The quantity of silicon which may be

present without having any very decided effect, depends very much on the relative proportion of carbon. Thus, with the carbon about 0·10 per cent., silicon might be present to the extent of 0·50 or more, without causing the metal to be brittle ; whereas with about 0·50 per cent. the same amount of silicon would cause the metal to be very red- and cold-short. Gautier states that metal containing only a trace of carbon may contain as much as 7 per cent. of silicon, and yet roll and weld. Metal containing much silicon cannot be coated with tin, owing to the silicon becoming oxidised during the operation and forming a protective coating of silica, which prevents the tin adhering. The presence of silicon may be detected during rolling by the behaviour of the scale, which either adheres to the rails or falls off in flat dry flakes, instead of leaving the metal freely and curling up, as is the case when there is only a trace of silicon.

Although, as we have seen, silicon under certain circumstances is an objectionable constituent of steel, yet in the production of sound castings it has proved itself a very valuable ally. The addition of silicon, in the form of highly silicious pig-iron containing a large percentage of manganese, to the molten metal just before it is run into the mould, has the effect of preventing the evolution of gas during the solidification of the metal, the silicon reducing the carbonic oxide whilst it is still held in solution in the metal, forming silica, which combines with any oxide of iron present. The manganese serves the double purpose of removing any excess of oxygen in the metal, and of rendering more fluid the silicate of iron produced by the action of the silicon on the carbonic oxide in the presence of oxide of iron. It is essential that oxygen should be kept out of the bath as much as possible at all stages of the process, to effect which a bath of spiegel is first formed, into which the metal is thrown, and then gradually melted. From time to time the amount of manganese present is determined by dipping a rod into the slag, and examining the colour of the fracture of the crust

formed on the rod. This estimation of the manganese forms a sufficient clue to the quantity of oxygen to be expected in the metal. These indications are, also, checked by direct tests on samples of the metal withdrawn for the purpose. Speaking generally, the bath is ready for the silicious and manganiferous pig-iron to be added when the metal-test indicates the same degree of softness that is required before adding the ferro-manganese in the ordinary open-hearth process. The metal remains perfectly tranquil after it has been run into the mould.

The character of the metal produced in this way is susceptible of considerable modification by special treatment after being cast. The cast metal, as it is taken from the mould, is on fracture found to be largely crystalline. By heating the metal to a cherry-red and cooling it more or less gradually, the grain becomes much more compact, the toughness and ductility are greatly increased, and the breaking-strain becomes slightly higher, whilst the proportion which the elastic limit bears to the breaking-strain is decreased. By reheating and cooling in oil the breaking-strain and the elastic limit are increased, whilst the toughness and extension are diminished. The fracture of the oil-cooled metal has a very fine texture.

Cast steel produced in this manner is characterised by great toughness, it being possible to bend it to quite as great an extent as the best wrought iron. It will have been seen that, within certain limits, the character of the metal can be varied, either by altering its composition or by modifying the treatment it receives after being cast. In general, the variation in breaking-strain does not exceed 6 tons per square inch. ranging from about 29 tons to 35 tons per square inch; the extension varies between 17 and 6 per cent. when tested on lengths of 8 inches, the standard length at present in this country. The elastic limit is found to range from $\frac{1}{2}$ to $\frac{5}{8}$ the breaking-strain. Stronger qualities can of course be made, but at the expense

of toughness. The working stress allowed to be put upon cast iron rarely exceeds 1 ton per square inch. Now it has been stated on authority that there are hydraulic riveters, made in this cast steel, which are in daily use with a working stress of 14 tons per square inch. Another great advantage possessed by this form of cast steel is, that it can be wrought under the hammer and welded with facility, not only to metal of a similar kind, but also to rolled steel and to wrought iron. The shrinkage of these steel castings is about $\frac{1}{4}$ inch per square foot. Young engineers would do well to study well this class of metal.

Sulphur.—This element is very active in causing red-shortness, even so small a quantity as one-tenth of one per cent. (0.10) being sufficient to make a rail roll badly. Its presence in such small quantity in steel used for tools and cutlery appears to be less objectionable, since it does not produce brittleness. The soundness of steel castings is impaired by sulphur.

Phosphorus behaves in exactly the reverse way to sulphur, producing cold-shortness. A given quantity of phosphorus has about the same effect in producing cold-shortness that the same amount of sulphur would have in causing red-shortness. It is somewhat singular that these two elements when present together tend to neutralise each other's effect. Phosphorus hardens iron to a greater extent than carbon, the ratio between them being at least as 2 to 1. The hardening effect of phosphorus is not, however, influenced by the rate of cooling, since the phosphorus cannot exist in the iron in two conditions—the combined and the uncombined—as is the case of carbon. As in the case of silicon, the amount of phosphorus which is admissible depends greatly upon the percentage of carbon. Thus, a rail might contain 0.25 of phosphorus, and yet not be dangerously brittle, provided carbon were nearly absent. But a rail containing only 0.10 per cent. of phosphorus and 0.30 to 0.50 of carbon would be useless. It is in steel intended for tools

and cutlery that phosphorus is more objectionable than in any other, as it renders it extremely difficult to temper, and a fine cutting edge cannot be obtained, if even so minute a quantity as twenty-five hundredths of one per cent. of phosphorus be present. In castings phosphorus in small quantity is beneficial (as already explained in speaking of foundry practice), owing to its causing the metal to cast sounder; and although the actual strength of the metal is slightly reduced, yet that of the casting as a whole is increased, and it is sharper in outline by reason of the greater fluidity of the metal.

Manganese.—This is the trusted physician of the iron and steel manufacturer. It is to this element that he looks for the cure of all the ills that iron is heir to, with the exception of the hereditary disease, 'phosphorus,' which experience has taught him must be dealt with by more modern treatment of a radical nature. Its uses have been so often referred to in the foregoing pages, that it is unnecessary to do more than summarise them here. Amongst the most important are the removal of oxygen from iron, the neutralisation of the deleterious effects of sulphur, and the production of a fluid slag, promoting its complete removal from the metal.

Copper has generally been considered to cause red-shortness in about the same proportion as it is produced by sulphur; but recent experiments of Herr Wasum appear to prove that with even 0·86 per cent. of copper, 0·23 of carbon, and 0·71 of manganese, ingot-iron will roll perfectly sound.

Chromium.—Steel containing a small quantity of this metal has a fine texture; a larger quantity produces hardness and brittleness. The application of this alloy has, so far, been very limited.

Chrome steel is stated to be manufactured at Brooklyn, America, in the following manner: the chrome-iron ore is finely powdered and reduced with charcoal in crucibles, the yield being about 45 per cent. of crude chromium, containing 30 per cent. of chromium and 3 per cent. of carbon.

This chromium-pig is granulated and mixed with a suitable quantity of pure wrought iron. Six charges of 70 lbs. each can be worked off in 24 hours. The percentage of chromium ranges between one-quarter and two.

Tungsten.—The effect of tungsten on steel is to greatly increase its hardness without the necessity of tempering ; in fact, tungsten steel cannot be tempered except by the use of oil, and then only with great difficulty, the richer alloys inevitably cracking. The great hardness of tungsten steel renders it very useful for tools for turning very hard metal. For such purposes there is an increasing demand for this alloy. The fracture of tungsten steel is remarkably fine-grained, and is very beautiful. The power of steel to retain magnetism is greatly increased by the presence of 2 or 3 per cent. of tungsten.

One word as regards the employment for cutlery of steel made by the Siemens and Bessemer methods. Cutlery manufacturers are apt to tell you that it is useless for the purpose ; nevertheless, on the Continent it is very largely used, and in this country to a considerable extent. It may be safely asserted that, with *proper care in the manufacture*, steel well suited for cutlery can be made both in the open-hearth and the converter. As already pointed out, the essential in the metal is, that it should not contain phosphorus, and, in respect to the manufacture, that the metal should be finally melted at a suitable temperature, and the greater part of the occluded gas got out.

In the foregoing pages it has been attempted to show in what progress has really consisted, and how it has been brought about. If we glance back for a moment, we see that the open-hearth processes embody the same principle as the first process by which steel was produced, viz. the mutual action of carburised iron and oxide of iron on one another, and the Bessemer process is, after all, though a fine offspring, and the natural descendant of the finery process, the origin of which, as we have seen, was due to modifications in the

primitive blast-furnaces. There is perfect continuity throughout, and what, after all, more natural?

Progress in the art of manufacturing iron and steel has been the joint work of the scientific chemist and of the engineer. As in the past, so in the future; success will depend on these two elements working harmoniously together.

COPPER.

The word copper has been derived from the corresponding Latin word *cuprum*. The Romans, who obtained their copper mainly from Cyprus, called it *æs Cyprium*; this term in time became shortened to *cyprium*, and finally to *cuprum*.

Native copper, *i.e.* copper in the metallic state, does not occur in Cyprus, nor, indeed, in any other part of the world, except in the Lake Superior district, in sufficient quantity to constitute a source of copper in itself. There is distinct evidence to show that the Romans reduced the ores of Cyprus (sulphides and carbonates) by methods which in principle are identical with our own.

Native Copper is sometimes found in masses of large size, having a very curious branch-like appearance, each branch being composed of crystals of copper, somewhat deformed, united together. Such masses have been obtained from the southern shore of Lake Superior. Some of the blocks weigh as much as 400 tons, and since the toughness of the metal prevents it from being blasted with gunpowder, much time and labour are expended in cutting the blocks into portable masses with steel chisels. Metallic copper is also found in veins disseminated in granite, in Cornwall and North Wales, and in many other parts of the world. A very remarkable form of native copper is the *copper sand*, or *copper barilla*, of Chili, which consists of grains of metallic copper mixed with quartz. Native copper, especially that from Lake Superior, is of a very pure description, and is tougher than

any but the best specimens of the copper extracted from the ores.

The following table exhibits the composition of the ores of copper :—

<i>Ores of Copper.</i>		Copper in 100 parts of pure ore
	Composition	
Red Copper Ore .	Copper, Oxygen . . .	89
Black Oxide . .	Copper, Oxygen . . .	80
Copper Glance .	Copper, Sulphur . . .	80
Indigo Copper .	Copper, Sulphur . . .	67
Copper Pyrites .	Copper, Iron, Sulphur . .	35
Peacock Copper .	Copper, Iron, Sulphur . .	56
Grey Copper Ore .	{ Copper, Iron, Sulphur, } { Antimony, Arsenic }	Variable
Malachite . . .	{ Copper, Oxygen, } { Carbonic Acid, Water }	58
Blue Malachite .	{ Copper, Oxygen, } { Carbonic Acid, Water }	55

Copper Pyrites or *Yellow Copper Ore* is the most abundant English ore of copper, being found in large quantities in Cornwall and Devon. It is also plentiful in Sweden, Saxony, Siberia, and Australia. The colour of pure copper pyrites is a fine brass-yellow, but some specimens are much paler, from the presence of iron pyrites. Copper pyrites is much softer than iron pyrites, and the richness of a sample may be in some measure inferred from this character. Although detached specimens of pure copper pyrites may be easily procured, it is always associated, in the vein, not only with the *vein-stone* or *gangue*, generally composed of quartz (silica) or fluor spar (fluoride of calcium), but with arsenical pyrites (composed of arsenic, iron, and sulphur) and tinstone (oxide of tin).

Peacock Ore or *Variegated Copper Ore* is found at St. Austle and Killarney. Like copper pyrites, it is composed of copper, iron, and sulphur, but it contains a larger proportion of copper than that ore.

Grey Copper Ore is one of the most abundant and important ores of this metal, as well as the most complex and variable in composition. Like the preceding ores, it con-

tains the copper in chemical combination with sulphur, but this latter element is also combined with iron, antimony, and arsenic, and generally with zinc and silver. The proportion of copper varies between 25 and 40 parts in the hundred, and the silver is very commonly present in sufficient quantity to render its extraction a matter of great importance. Cornwall and Freiberg furnish large supplies of grey copper ore.

Copper-glance, also a Cornish ore of great importance, is a chemical compound of copper and sulphur which is generally free from any important foreign minerals.

Indigo Copper, so named from its dark blue colour, is found in Chili.

Red Copper Ore differs from the preceding ores in being free from sulphur, and since it is found pretty abundantly in Cornwall, Cuba, and elsewhere, it plays a prominent part in some of the stages of the process of copper-smelting.

Black Oxide of Copper is found in Chili.

Malachite or *Green Carbonate of Copper* is a very fine green ore, some specimens being so beautifully veined that they are more highly prized for ornamental purposes than as an ore of copper. It is a very pure and valuable ore, but is not abundant in England, being found chiefly in Siberia, the Ural Mountains, and Australia.

Blue Malachite or *Azurite* or *Blue Carbonate of Copper* contains a larger proportion of carbonic acid than the green carbonate, and is generally found in the same localities. The mines of Burra Burra in South Australia are noted for malachite ores, which yield copper of excellent quality.

The Cornish copper ores are shipped to Swansea in order to be smelted, the vessels returning to Cornwall laden with the coal required for the tin-works. The ores from Australia, Chili, Cuba (*Cobre ores*), &c., are also received at Swansea, the neighbourhood of which furnishes an abundance of coal. These latter ores are also to a considerable extent smelted in the countries in which they occur.

Our market is now ruled by the price of Chili and

Australian copper, and not by the standard fixed by the Swansea smelters, as formerly. Copper ores are also mined in Anglesea, the Isle of Man, Lancashire, and some parts of Ireland and Scotland.

EXTRACTION OF COPPER FROM ITS ORES.

Probably no other metallurgic operation presents such an appearance of complexity as the smelting of copper ores, but this is due to the great variety of the ores to be treated, which necessitates their introduction at different stages of the process. Thus, a smelting process adapted for copper pyrites must contain provisions for the removal of arsenic and sulphur, which are not present in the carbonates and the oxides of copper, so that the processes of smelting are arranged in such a manner that these ores, as well as the slags obtained in some of the operations, can be introduced after the sulphur and arsenic have been expelled.

In a work like the present, it is not advisable to attempt a detailed account of smelting processes which are subject to frequent alterations in order to suit different lots of ore, particularly when such alterations result from the application of practical experience on the part of the smelter, and do not admit of clear explanation upon simple chemical principles. A general outline only of the extraction of copper from its ores will be given here, and before this is entered upon, it may assist the reader to state that it may be summed up under the following heads :

1. *Roasting processes*, intended to expel arsenic and sulphur, and to convert the iron into oxide of iron.
2. *Melting processes*, intended to remove the oxide of iron by dissolving it with silica at a high temperature, and to obtain the copper as a pure combination of copper with sulphur (sulphide of copper).
3. Roasting and melting, in a single process, to expel the sulphur and obtain metallic copper.

Before being subjected to the first process, the ores are

broken into pieces of the size of a nut, and so assorted that the lot to be smelted may contain about eight or ten parts of metallic copper in the hundred. Moreover, as there is much *gangue* or earthy matter associated with the ores, they are, if possible, so mixed that they may serve as *fluxes* to each other, by producing chemical compounds capable of becoming liquefied by the high temperature of the furnace.

The *fluor* spar*, which is so commonly associated with copper pyrites, derives its name from its power to effect the liquefaction of earthy substances. Fluor spar is composed

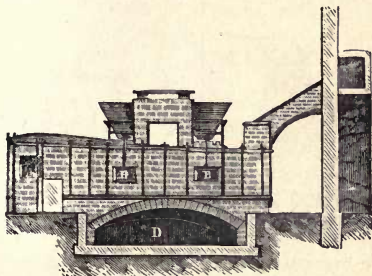


FIG. 55.—Furnace for roasting Copper Ores. BB, Working doors.
D, Vault for receiving the roasted ore.

of calcium and fluorine; if it be strongly heated in contact with silica (quartz), which consists of oxygen combined with silicon, the latter takes up the fluorine to form fluoride of silicon gas, whilst the calcium and oxygen unite to produce lime, which combines with another portion of the silica to form a silicate of lime. The silicate of lime would not easily fuse into a slag by itself, but when clay and oxide of iron are present, as is always the case in the melting furnaces, a slag is readily produced.

1st Process in Copper-smelting. Calcining or Roasting to Expel Arsenic and part of the Sulphur.—The roasting-furnace

* From the Latin *fluere*, to flow.

or *calciner* (Figs. 55, 56, 57) is a *reverberatory* furnace, with a hearth (A) of large size (about sixteen feet by fourteen) to allow of the ore being spread out in a thin layer upon it. The hearth is commonly built of fire-bricks set on edge and bedded in fire-clay, and the flame is reverberated upon it by an arch of about two feet in average height. At one end of the hearth, near the fire-place, there is an opening or flue (o) through which air may be admitted to the hearth, to furnish the oxygen necessary for the chemical changes effected in the roasting process. On each side of the hearth there are

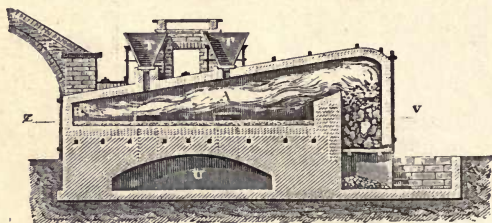


FIG. 56.—Furnace for roasting Copper Ores. Section through the line x y of the plan (fig. 57).

two openings (r) closed with iron doors, through which the roasted ore is raked out into the arch (u) beneath the furnace. The ore is admitted by opening the hoppers (t) over the arch of the furnace, where it is previously warmed by the waste heat. The fuel employed in the calciners at Swansea is non-caking mixed with one-fourth of bituminous or caking coal, which is necessary to counteract the tendency of the coal to split up into small pieces and fall through the grate unburnt, the bituminous coal being softened by the heat, and binding the free-burning together. The fire of the calciners requires special management in order that the ore upon the hearth may be brought to the proper temperature. Both the free-burning and caking coal used have had the lumps screened out for household purposes. The

small coal would easily fall through the grate. To avoid this, a layer of *clinker* or fused ash from the coal is built up on the bars of the grate (F), preserving them at the same time from direct contact with the glowing coal, and air-passages are made through this layer, so that the air becomes heated in passing through it, before actually reaching the fire, the combustion of the fuel being thus effected by a current of heated air. The oxygen of the air, passing through

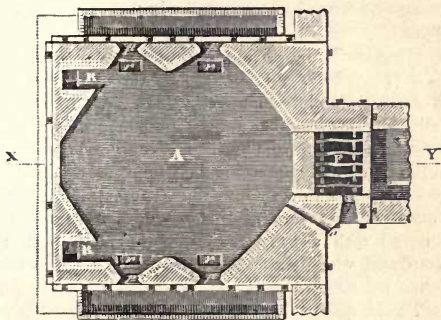


FIG. 57.—Furnace for roasting Copper Ores. Plan at the line *z v* of the section (fig. 56).

the column of heated fuel, combines with the carbon to form carbonic oxide (see p. 5), and this gas, being highly heated, takes fire in the air admitted on to the hearth of the furnace, giving a sheet of flame which is drawn through the furnace by the action of the chimney with which the flues (R) communicate, and raises the ore to the temperature necessary for roasting it. Since the air is heavier than the burning gas, a layer of air always exists beneath the latter, separating it from the ore, thus preventing the ore from attaining its melting point, and securing a sufficient supply of oxygen.

Each calciner is charged with three tons of the broken ore, which is spread evenly over the hearth, and roasted for twelve hours, being occasionally raked over through the

working-doors (*p*) in order to expose fresh portions to the action of the air, and to prevent any part of the ore from being melted. At this high temperature, the arsenic present in the copper ore combines with oxygen from the air to form *arsenious acid* (white arsenic) which passes, in the form of vapour, into the flues. About half of the sulphur in the ore also combines with oxygen to form *sulphurous acid* gas which passes up the chimney, a small quantity of *sulphuric acid* being also formed and remaining in the ore as sulphate of copper.

Since iron exerts the greater chemical attraction for oxygen, and copper for sulphur, a large proportion of iron acquires oxygen and becomes converted into an *oxide of iron*, while a much smaller proportion of the copper combines with the oxygen from the air to form *suboxide of copper*. When the gases and vapours issuing from the calciners are allowed to escape directly into the air, they form a dense grey cloud of *copper-smoke* which contains the sulphurous acid, mixed with a little vapour of sulphuric acid, the arsenious acid, which condenses in the air to a fine powder, and some hydrofluoric acid gas, produced from the fluor spar. The injurious effect of these products upon the health and vegetation of the neighbourhood has induced the copper smelters to devise means for condensing them by passing them into flues and condensing chambers where they are met by showers of water.

At some works it has been found profitable to convert the sulphurous acid into oil of vitriol instead of allowing it to escape, but in this case it is necessary to prevent the products of combustion of the fuel from mixing with the copper-smoke. *Spence's calciner* employed for this purpose has the fire passing under the hearth instead of over it. This furnace is 50 feet long, and the ore is gradually raked from the cooler to the hotter end as it becomes less fusible. The waste heat of an adjoining smelting furnace is sometimes employed in these calciners, and the calcined ore is raked at once into the smelting furnace. In *Gerstenhöffer's furnace* the ores are

crushed between rollers, and allowed to fall over rows of red hot bricks in a vertical furnace, through which a blast of heated air is passed in order to burn the sulphur into sulphurous acid, which is then conducted into the leaden chambers, where it is converted into oil of vitriol.

2nd Process in Copper-smelting. Melting for Coarse Metal, to Dissolve the Oxide of Iron as a Silicate.—It has been seen that the 1st process has had the effect of converting a large proportion of the sulphuret of iron present in the pyrites into oxide of iron, which it is the object of the present process to remove by causing it to combine with silica, to form a compound capable of being melted and separated from the rest of the ore. At this stage the copper ores containing silica (quartz) can be introduced with advantage, provided that they are free from sulphur. It must not be forgotten that, during the process of calcining, a small proportion of the sulphuret of copper in the pyrites has been converted into an oxide of copper, which resembles the oxide of iron in its property of combining with silica at a high temperature, to form a melted silicate which would pass away in the slag, entailing a considerable loss of copper. This is prevented by the sulphuret of iron which is still present in the calcined ore, and, at the high temperature at which the fusion is effected, exchanges constituents with the oxide of copper, forming oxide of iron and sulphuret of copper. The slag from the 4th process, to be presently described, is also appropriately introduced in this fusion, since it contains a considerable quantity of oxide of copper, which exchanges, as above, with the sulphuret of iron in the calcined ore, furnishing more sulphuret of copper to pass into the coarse metal, and oxide of iron to be removed in the slag. The slag from the 4th process (called *metal slag*), being basic, assists in fluxing the silica in the ore. In some cases, the smelter adds some fluor spar in order to facilitate the fusion of the charge.

The *ore-furnace* (Figs. 58, 59), as it is called, in which the melting for coarse metal is effected, is also a reverberatory

furnace, but its hearth (A) is much smaller than that of the calciner (usually about one-third of the size), because the charge has to be raised to a much higher temperature; for which reason, also, the fire-grate is larger in proportion; the hearth is also slightly inclined on all sides towards a depres-

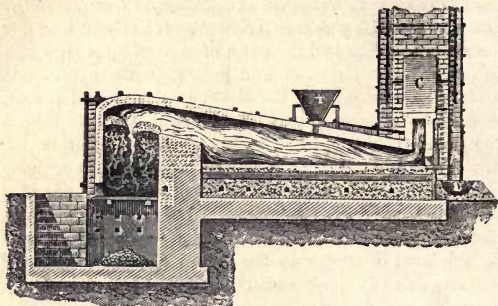


FIG. 58.—Section of Ore-furnace for smelting Copper Ores. T, Hopper for introducing the charge. P, Tap-hole for discharging the slag into the slag-moulds U. C, Flue leading to the chimney.

sion or cavity (B) at one side, which serves as a crucible in which the melted coarse-metal collects. The fuel is a mixture of free-burning with one-third of bituminous coal. The charge of this furnace is composed of the following materials, selected for the reasons above given, viz. :—

Calcined or roasted ore, usually about 18 cwt.

Ores containing oxide of copper and silica, 3 cwt.

Metal-slag from process 4, containing oxide of iron, silica, and some oxide of copper, 6 cwt.

Fluor-spar, occasionally.

The slag is the first to fuse, in about half-an-hour after the charge has been introduced, and by degrees the whole of the materials become liquid, and enter into violent ebullition, caused by disengagement of sulphurous acid gas, produced by a secondary decomposition of no importance from a

metallurgic point of view, save that the ebullition favours the intimate mixture of the melted matters on the hearth.

After three or four hours, the furnace-man mixes up the melted matters with a rake, and raises the temperature very considerably, to favour the separation of the coarse metal from the slag. In about half-an-hour, the tap-hole (*b*, Fig. 59), which communicates with the cavity in the hearth, is opened, and the *matt** or *regulus* of coarse metal is run out,

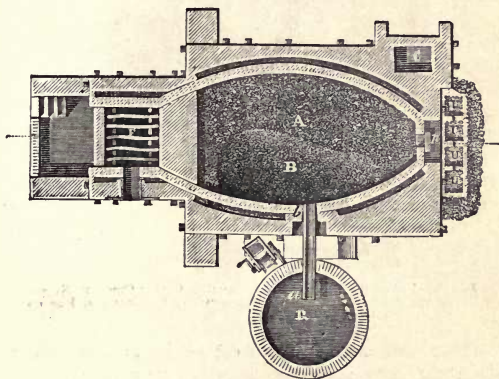


FIG. 59.—Plan of Ore-furnace for smelting Copper Ores. F, The grate.
R, Tank for granulating the coarse metal.

through an iron gutter (*a*) into an iron box (*G*, Fig. 60), perforated at the bottom, and standing in a cistern through which water is constantly running; the coarse metal is thus *granulated* or divided into small irregular grains, in order to fit it for undergoing the next operation.

Sometimes the regulus from two or three operations is allowed to accumulate in the furnace before tapping, the slag alone being raked out before the introduction of a fresh charge.

The iron box containing the regulus is raised from out of

* From the French *mat*, heavy.

the cistern by a winch (w), and its contents are carried to the calcining furnace.

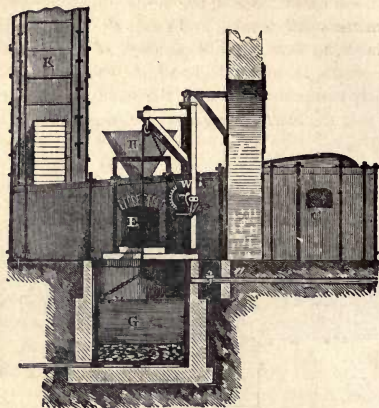


FIG. 60.—Elevation of Ore-furnace for smelting Copper Ores. H, Hopper for introducing the charge. K, Chimney. C, Fire-door. a, Pipe for supplying water to the tank.

This coarse metal contains copper, iron, and sulphur in about the same proportion in which they are present in pure copper pyrites, so that the copper amounts to about 33 parts in the hundred, or nearly four times the proportion contained in the raw ore at the commencement of the process.

The slag (*ore-furnace slag*) is raked out into sand-moulds (U, Fig. 59), connected with each other by openings in their sides, where it solidifies into blocks of a black, somewhat glassy, appearance, interspersed with white fragments of quartz. It is used for rough building purposes in the neighbourhood of the copper works. The ore-furnace slag is composed essentially of oxide of iron (ferrous oxide) and silica combined in about equal proportions, and would be spoken of, in chemical language, as a *silicate of iron* or

ferrous silicate. It contains also a little copper, usually amounting to one part in 140 parts, representing a loss to the smelter which appears unavoidable. Occasionally, a small quantity of regulus is found at the bottom of the blocks of slag, from which it is separated by hand-picking. Fig. 61 exhibits the general arrangements connected with the ore-furnace, and shows the furnace-man discharging the slag.

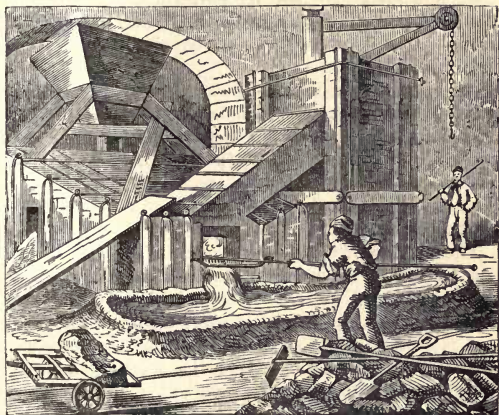


FIG. 61.—Copper Smelting-furnace.

3rd Process in Copper-smelting. Calcination of the Coarse Metal, to convert more of the Sulphuret of Iron into Oxide.—Now that the earthy matter has been removed in the slag, it is far easier to oxidise the sulphuret of iron than it was in the first calcining process. To effect this, three tons of the granulated coarse metal are roasted in the calcining furnace (Fig. 57) for 24 hours, the temperature being moderated at the commencement, to avoid fusion, and gradually raised in proportion as the removal of the sulphur diminishes the fusibility of the charge, which is raked over every two hours.

About one-half of the sulphur is converted by the oxygen of the air into sulphurous and sulphuric acids, which escape in vapour, another portion of oxygen combining with the iron from which the sulphur has been removed, to form oxide of iron, so that the roasted coarse metal consists essentially of sulphuret of copper, oxide of iron, and some unchanged sulphuret of iron.

4th Process in Copper-smelting. Fusion of the Calcined Coarse Metal to remove all the Iron and to obtain Fine Metal.—The principles involved in this process are the same as in the second process.

The fusion is effected in a furnace which does not differ materially from that employed in the 2nd process, except that there is no cavity in the hearth, which is made to slope from all parts towards the tap-hole (Fig. 59). The charge consists of—

Calcined coarse metal (about one ton)

Roaster-slag from the 5th process

Refinery-slag from the 6th process

Ores containing oxide and carbonate of copper

} About
12 cwt.

(The roaster and refinery slags contain silica in combination with the oxides of iron and copper.)

These materials are fused together for about six hours, when they divide, as before, into a regulus or matt, and a slag, which remains above it. This regulus is called *fine metal*, to distinguish it from the *coarse metal* of the 2nd process ; it may contain from 60 to 80 per cent. of copper, according to the amount of oxidised products and ores containing oxide and carbonate of copper added to the melting-charge. The different qualities are distinguished by specific terms ; thus, when the metal contains from 60 to 70 per cent. of copper and has a smooth shining fracture and blue colour, it is called *blue metal* ; from 75 to 78 per cent., the fracture is granular, the lustre greasy, and the colour greyish-white, it is then called *white metal* ; when the percentage of copper is above 78, the surface of the metal is covered with pimples, and *moss copper* is found in the air cavities, it is then called *pimple metal*. The pimples are formed by escaping sulphurous acid gas.

When it is intended to manufacture *best selected copper* for making brass, gun-metal, &c., the fine metal is made to undergo a partial roasting ; the various impurities which are present tend to collect in the metallic copper, which is thus separated from the melted mass of regulus. Two or more roastings may be required. The metallic copper containing the impurities is termed *bottoms*. The unreduced fine metal or *regule*, which should now be nearly free from impurities, is treated in the ordinary way for copper. If gold, silver, tin, lead, iron, nickel, manganese, antimony, or arsenic be present in only small traces, they can, without difficulty, be entirely eliminated by the above selecting process.

The composition of a sample of these bottoms is here given, in 100 parts : copper 74, tin 14, antimony $4\frac{1}{2}$, lead 1, iron $2\frac{1}{2}$, sulphur 4. It is evident that the metallic copper which has separated has decomposed the sulphurets of tin, antimony, &c. contained in the metal, and has combined with those metals to form an alloy, which is heavier than the metal and sinks to the bottom.

In some smelting-works, where the fine metal is not obtained in so pure a condition, and contains only 60 parts of copper in the hundred, it is again submitted to the two processes of calcining and melting, exactly as in processes 3 and 4, when it yields *black copper* or *coarse copper*, which contains from 70 to 80 parts of copper in the hundred.

The *metal-slag*, as the slag from the 4th process is termed, presents an appearance very different from that of the ore-furnace slag ; it is very crystalline and lustrous, and consists chiefly of oxide of iron combined with silica, but it contains a considerable proportion of copper, partly in the form of an oxide in combination with silica, and partly as small particles of metallic copper, disseminated through the mass. In some specimens of the metal-slag, the copper appears in very fine brilliant filaments, forming *copper-moss*. This slag is usually employed as part of the charge in the 2nd process (melting for coarse metal).

5th Process in Copper-smelting. Calcining or Roasting the Fine Metal to remove Sulphur and obtain Blister-Copper.—

The manner in which this process is carried out is varied according to the degree of purity of the fine metal, but the chemical principles which it involves are the following: When a compound of copper with sulphur is heated in air, the sulphur combines with the oxygen of the air, and is thus gradually removed in the form of sulphurous acid gas, the copper also combining with oxygen, and being left as oxide of copper. Further, when an oxide of copper (or compound of copper with oxygen) is melted in contact with a sulphuret of copper (or compound of copper with sulphur), the oxygen of the former combines with the sulphur of the latter to form sulphurous acid gas, and the copper is separated in the metallic state.

The pigs of blue metal are introduced, to the amount of $1\frac{1}{2}$ ton, into a reverberatory furnace, where they are roasted, at a gradually increasing temperature so as to avoid fusion, for about four hours, in order that a part of the sulphuret of copper may be converted into oxide of copper. When it is judged that this has been effected to a proper extent, the temperature is further raised so as to fuse the materials upon the hearth, the doors of the furnace being closed in order to avoid excess of air. As soon as the mass is fairly liquefied, the temperature is somewhat reduced, being again raised towards the close. During this fusion a violent effervescence is observed in the liquid mass, due to the escape of sulphurous acid gas, formed by the union of the sulphur from the sulphuret with the oxygen from the oxide of copper, whilst metallic copper subsides, in a fused state, and is run out into sand-moulds, where it solidifies into ingots, which preserve a blistered appearance, caused by the escape of sulphurous acid during solidification. The duration of the process depends upon the degree of purity of the blue metal under treatment, but it varies between 12 and 24 hours.

A small quantity of slag (called *roaster-slag*) is formed during the fusion, which resembles pumice in its porous

texture, but has a dark red-brown colour, and consists of the oxides of iron and copper combined with silica derived partly from the hearth of the furnace, and partly from the sand-moulds in which the ingots of blue metal are cast. This slag contains about 16 parts of copper in a hundred, and is used as a portion of the charge in the 4th process.

The roasting-furnace employed in this process is often constructed with an air-channel (Fig. 62) traversing the whole length of the fire-bridge, open to the air at both ends, and communicating with the hearth of the furnace through two openings (*b b*) in the brickwork. This permits the introduction of heated air into the hearth, by which the roasting is much facilitated.

6th Process of Copper-smelting. Refining and Toughening, to purify the Copper.—The pigs of blister-copper are far from pure; they contain considerable proportions of sulphur, arsenic, iron, tin, lead and other foreign substances, varying according to the descriptions of ore employed

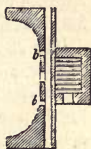


FIG. 62.

In order to remove these impurities, the oxygen of atmospheric air is brought into use. The furnace employed does not differ very materially from the melting-furnace used in the 2nd process (Fig. 58). The blister-copper to be refined is piled, in charges of 6 or 8 tons, upon the hearth, in such a manner as to allow air to circulate freely among the ingots. A moderate heat is applied at first, to allow the oxygen of the air to act upon the blister-copper, an action which is facilitated by the porous structure of the metal. The sulphur then becomes converted into sulphurous acid gas, and the arsenic into arsenious acid, which passes off in vapour, whilst the iron, tin, lead and other foreign metals are converted into oxides, as well as a portion of the copper. After being roasted for about six hours, the metal is melted, when a thin layer of slag is formed upon its surface; after raking this off, a large sample of the copper is withdrawn and examined by the refiner, who can judge from the appearance of its surface if the oxidation has

proceeded to the necessary extent. In order to toughen the metal, its surface is covered with wood-charcoal or anthracite, which is renewed from time to time, so as to shield the copper from further oxidation, and the melted metal is stirred with a pole of young wood (usually birch), until a small sample half cut through with a chisel and then broken exhibits a fine close grain, a silky fracture, and a light-red colour, and a small ingot, cast for the purpose and hammered when red-hot, is found to be soft and free from cracks at the edges. The copper is then said to be at *tough-pitch*, and is taken out in iron ladles lined with clay, and cast into ingots of *tough-cake* copper.

The effect of this process of *poling*, as it is termed, in toughening the copper, depends upon the removal of oxygen from the metal. When the blister-copper has been refined, as above described, by being very slowly melted in contact with air, it is found to have taken up a large proportion of oxygen, which is contained dissolved in the metal as an oxide (*suboxide*) of copper. The presence of the oxygen, though it does not amount to more than two or three parts in a thousand of copper, has the effect of rendering the copper brittle or *dry*, so that a small ingot of it is easily broken when hammered, and its fracture exhibits a deep red colour, and a coarse-grained, somewhat crystalline structure. When the pole is plunged beneath the melted metal the combustible gases, generated from the wood by the heat, effect the removal of the oxygen from the metal, and bring it by degrees to tough-pitch. If, during the operation of casting the ingots, the surface of the metal on the hearth be not well covered with charcoal or anthracite, the copper will go *back* or become brittle again, in consequence of the absorption of oxygen from the air.

If the process of poling be continued after the copper has been brought to tough-pitch, it becomes even more brittle than before it was poled, an effect which was formerly ascribed to the combination of the copper with a little carbon from the wood ; but since analysis has failed to prove

the presence of the carbon, the following less simple explanation, based upon experiment, is now generally received. Perfectly pure copper exhibits the malleability and ductility of the metal in the highest perfection, but these qualities are deteriorated by the presence of small proportions of the various foreign matters, such as sulphur, tin, antimony, &c., which cannot be entirely removed in the refining process. The injurious effect of these impurities, however, is counteracted in some measure by the presence of a small proportion of oxygen (not exceeding two parts in a thousand), so that if this element be entirely removed, the copper will be *overpoled*, exhibiting a brittle character, due to some of the above-named impurities. On the other hand, if too much oxygen has been left in the metal, the copper is dry or *underpoled*. The effect of overpoling upon the metal may be remedied by allowing air to act for a short time upon the melted copper, so that a small quantity of oxygen may be absorbed by it.

When the copper is intended for rolling into sheets, it is usual to add lead, in the proportion of about five parts to a thousand of copper, just before skimming the surface in order to ladle out the copper. The metal is well stirred after the addition of lead, in order that the action of the air may produce an oxide of lead, which combines with the oxides of tin, antimony, and other foreign metals, to form a liquid slag, which rises to the surface of the metal and is skimmed off before casting. It is necessary that the removal of the lead from the copper by oxidation should be as complete as possible, since its presence would prevent the scale of oxide of copper from being easily detached from the sheet during the process of rolling, and even $\frac{1}{10}$ th part of lead in 100 parts of copper suffices to injure its quality.

This treatment of the metal with lead is called *scorification*, from the *scoria* or slag which forms upon the surface.

The *refinery slag*, skimmed from the surface of the melted copper before commencing the process of poling, has a dull brown-red colour, with a purple shade, and consists almost entirely of an oxide of copper (suboxide) combined with

silica derived from the hearth and from the sand-moulds employed to cast the blistered copper. It is employed in the 4th process (fusion for fine metal).

The hearths of the copper-furnaces become strongly impregnated with copper in course of time, and are broken out in order that the metal may be removed from them.

The following modified method has been found to work successfully. By calcination or mixture a 28 per cent. coarse metal is produced. With this percentage a cleaner slag is obtained than with a 35 per cent.; a 35 per cent. has, however, the advantage of calcining better in the subsequent operation. In order to obtain this latter advantage as well as a clean slag, the 28 per cent. metal is melted down with the requisite amount of roaster and refinery slag, thus enriching the metal to the required extent. This method has the additional advantage that the roaster and refinery slags being introduced at an earlier stage than in the ordinary process, the impurities have a greater chance of being eliminated.

Extraction of Copper from the Bituminous Schists of Mansfeld.—Although most of the copper sent into commerce is extracted by the Welsh process, other methods are sometimes followed on the Continent for the treatment of poor ores, especially when coal is not abundant, for the coal required for the Welsh process amounts to eighteen times the weight of the copper. Thus, at Mansfeld, an ore is extensively worked which contains not more than four parts of copper in a hundred, in the form of crystals of copper pyrites, diffused through a clay slate containing a large proportion of bituminous matter. The consumption of fuel in extracting the copper from this ore is only one-third of that in the Welsh process.

The ore is first roasted in large heaps made up with alternate layers of brush-wood, the bituminous matter also serving as fuel. A heap containing 200 tons of ore will go on burning for fifteen or twenty weeks. In this process, which corresponds to the first calcination in the Welsh method, a part of

the sulphur passes off as sulphurous acid, and much of the iron is converted into an oxide.

The next process is similar to the Welsh fusion for coarse metal, and consists in melting the roasted ore with some fluor spar, to serve as a flux, some copper-ore containing carbonate of lime, for the same purpose, and some slags containing oxide of copper to decompose the sulphuret of iron and remove the iron as a silicate (page 235). The fusion is not conducted in a reverberatory furnace, as in the Welsh process, but in a small blast-furnace (Figs 63, 64), about 14 feet high, and 3 feet in its greatest width. The blast is supplied by two tuyères (*t*) placed side by side, about 2 feet above the bottom of the furnace, from which the melted matters are conducted through two channels (o o') into two basins (c c') about 3 feet in diameter and 20 inches deep, lined with a mixture of clay and charcoal dust; when one of these basins is filled, the channel communicating with it is closed, and the melted matters from the furnace are run into the other basin. The furnace is provided with a chimney (G) 30 or 40 feet high.



FIG. 63.—Blast-furnace employed for smelting the Bituminous Schists at Mansfeld.

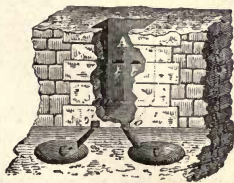


FIG. 64.—Hearth of Blast-furnace employed at Mansfeld.

The fuel employed is either charcoal or a mixture of charcoal and gas-coke, which is charged alternately with the ore, as in an iron blast-furnace. The chemical changes which take place in the furnace resemble those in the Welsh process of melting for coarse metal, and the liquid matter which flows into the

receiving basins divides into two portions, the lower layer consisting of the sulphurets of copper and iron, and the upper layer of slag composed chiefly of silicate of iron containing but little copper. The slag is ladled out into moulds and employed for building. The *matt*, as the lower layer is called, is removed in crusts, as it solidifies.

If the matt contains less than thirty parts of copper in the hundred, it is again roasted and treated as before, so as to remove more of the sulphuret of iron; but if it contains

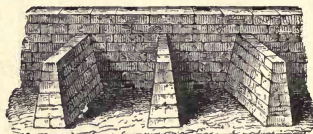


FIG. 65.—Roasting-stalls employed at Mansfeld.

more than this proportion, it is at once roasted in a special open furnace (Fig. 65), which consists of six separate compartments or stalls with flues running up the back walls in

order to create a draught. The matt is placed upon a wood fire in the first compartment, which is then closed by building up a temporary wall; when it has been calcined here for a certain time, it is transferred to the second compartment, and then to the third. It is now introduced into a wooden vessel and washed with water in order to dissolve the sulphate of copper which has been formed by the combination of the oxygen of the air with the sulphuret of copper. The washed matt is roasted again in the fourth, fifth, and sixth compartments, in succession, being treated with water after every roasting. The solution of sulphate of copper thus obtained is evaporated and crystallised, yielding *blue vitriol*, which is sent into commerce.

This operation of roasting, which lasts seven or eight weeks, corresponds to the *calcination of the coarse metal* in the Welsh process.

The roasted matt, containing oxide of iron and sulphuret of copper, is treated as in the *melting for fine metal*, being fused with siliceous slags which dissolve the oxide of iron.

The fusion is effected in a blast furnace similar to that described above, but of smaller dimensions. The liquid matter in the receiving basins divides into three layers, the uppermost consisting of slag, the middle layer of a matt containing 60 or 70 parts of copper in the hundred, combined with sulphur (representing the fine metal of the Welsh copper smelting), which is again roasted and smelted, and the lowest layer of *black copper*, which consists of impure metallic copper, containing about 95 parts of copper in the hundred, with 3 or 4 parts of iron, 1 part of sulphur, and sometimes as much as 120 ounces of silver to the ton. When a sufficient quantity of silver is present to pay for extraction, the black copper is subjected to a process for that purpose, which will be described under silver, and is afterwards refined.

The refining of the black copper, after separating the silver, is conducted in a reverberatory furnace (Figs. 66, 67), the hearth (A) of which is lined with clay and powdered charcoal, upon which the black copper is melted with the flame of a wood fire in the grate (F) and air is thrown upon its surface through two tuyères (t), when the oxygen of the air removes the sulphur as sulphurous acid, and converts the foreign metals into oxides which collect as a slag upon the surface. When the refining is nearly completed, a red slag containing much red oxide of copper forms, and a small sample is withdrawn and carefully examined by forging. If it be deemed sufficiently pure, the copper is run out, through the openings (o), into receiving basins (B) and removed in *rosettes* by throwing water upon it, and taking off the films of metal thus solidified. The rosettes thus obtained consist of *dry copper*, containing too large a proportion of red oxide. It is refined in the *German hearth* (Figs. 68, 69) which consists of a basin (c) about 16 inches wide, lined with a mixture of clay and powdered charcoal, and furnished with a blast-pipe (τ) like that of a blacksmith's forge. The copper being melted in this basin, is covered with charcoal and kept fused until the copper is at tough-pitch in consequence of the reduction of a sufficient proportion of the red oxide of

copper, the same attention and judgment being necessary as in the Welsh process of poling (page 241). The refined metal is then ladled into ingot-moulds.

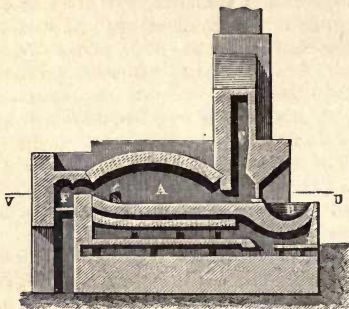


FIG. 66.—Section of Furnace for refining Black Copper, at Mansfeld, made at the line x of the plan, Fig. 67.

When the black copper is not rich enough to be treated for silver, it is at once refined in the German hearth just

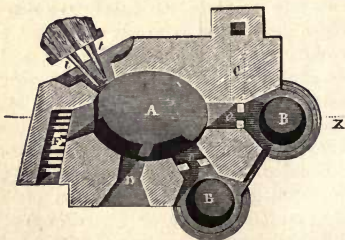


FIG. 67.—Plan of Furnace for refining Black Copper, at Mansfeld, made at the line v u of the section, Fig. 66.

described, but the process is conducted on the principle of the refining of blistered copper in the Welsh process, the impurities being oxidised by the air from the blast-pipe.

The hearth being filled with glowing charcoal, the black copper is placed upon it and gradually fused in the blast, so that the sulphur may burn off as sulphurous acid, and the foreign metals may be converted into oxides which are run off by a channel (*i i'*) provided for them. Alternate charges of black copper and charcoal are added from time to time, and the process continued until the workman perceives, by the inspection of a small sample, that the operation of refining is completed, when the surface is skimmed, and the copper removed in rosettes which are afterwards toughened as described above. The rosettes first removed are sometimes

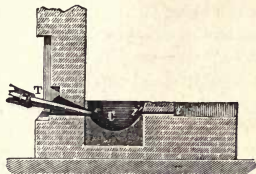


FIG. 68.



FIG. 69.

German Hearth used for refining Rosette Copper, at Mansfeld.

rich in nickel, and are subjected to a special treatment for the extraction of that metal, which is valuable in itself and injurious to the quality of the copper.

Precipitate Copper from Pyrites.—A very considerable quantity of copper is extracted from the residues after roasting pyrites (chiefly Spanish) at the alkali works for the sake of the sulphur, which is converted into oil of vitriol. These residues contain 2 or 3 parts of copper in 100, with very small quantities of silver. They are mixed with common salt (chloride of sodium), and moderately heated in a reverberatory furnace, when the copper and silver are converted into chlorides of those metals. On washing the roasted mass with water, these chlorides are dissolved, for the chloride of silver, though insoluble in water, dissolves in a solution of common salt. The silver is separated from the solution by adding some iodide of zinc, which precipitates

the silver as iodide of silver, and the copper is afterwards obtained by putting in some scrap iron, which combines with the chlorine and causes the separation of metallic copper, to be afterwards melted. The iodide of silver is placed in contact with metallic zinc, which separates the silver, combining with the iodine to form iodide of zinc, which is employed to precipitate the silver from a fresh batch of the material.

VARIOUS DESCRIPTIONS OF COMMERCIAL COPPER.

Cement Copper is extracted from the water which is pumped out of copper mines, or is collected from waste heaps, called *blue water*, its colour being due to the presence of sulphate of copper, produced by the action of the air upon the combination of copper and sulphur present in the ore. This water is pumped into tanks containing scrap iron, which gradually enters into solution as sulphate of iron, the copper being deposited in the metallic state. The fine red colour of the copper, and the pale green of the sulphate of iron, give the contents of the tanks a very beautiful appearance, especially in the sunshine. The *copper of cementation* thus produced, being almost chemically pure, is of very excellent quality. At Schmöllnitz in Hungary, and in the Island of Anglesea, cement copper is largely prepared in this manner.

Rosette Copper or *Rose Copper* is made chiefly at Chessy in France, by throwing water upon the surface of the melted copper, and removing the solidified metal in films, which have a beautiful red coating of an oxide (suboxide) of copper, formed by the action of the oxygen of the water upon the metal. These rosettes are plunged into water as soon as they are removed, for if they were allowed to cool in the air the further oxidation would produce a thick scale on the surface, spoiling the colour of the copper.

Japan Copper resembles the preceding in colour, and is cast into ingots weighing only six ounces each, for exportation to the East Indies. It is coloured by being cast under warm water on canvas.

Copper is sometimes cast into thin plates by pouring into

the mould enough metal to form a single plate, which is allowed to cool before pouring in a fresh quantity, when a film of suboxide of copper is formed upon the surface of the first plate, which prevents it from adhering to the next, so that the plates are easily separated when the moulding-case is removed.

Bean-shot Copper is made by pouring melted copper through a perforated ladle into a vessel of hot water, when it forms round fragments like shot, which are very convenient for the manufacture of brass. When cold water is used, the metal is obtained in flakes, which are termed *feathered shot*.

In order to remove the scale of oxide from rolled copper plates before sending them into the market, the somewhat inexplicable course is adopted of washing them with urine, then heating them in a reverberatory furnace and plunging them while hot into water, when the scale detaches itself. The sheets are then smoothed between rollers.

Effect of the presence of Foreign Matters upon the quality of Copper.—From the circumstance that the refiner tests the quality of copper by forging a *hot* sample, it will be inferred that the effect of impurities upon its malleability and tenacity is more perceptible at a high than at a low temperature. The foreign matters which commercial copper is liable to contain are arsenic, sulphur, antimony, tin, bismuth, lead, silver, iron, and nickel. Of these, sulphur and antimony are generally considered the most injurious in diminishing the malleability and tenacity of the metal. Arsenic is very commonly found in copper, amounting, in some of the Spanish coppers, to as much as one part in a thousand, and was formerly supposed to be as injurious to the quality of the copper as antimony is, but modern experience has shown that copper may be easily rolled and drawn into wire even when it contains a considerable proportion of arsenic. A small proportion of tin is believed to increase the toughness of copper, but bismuth and nickel have the opposite effect.

The conducting power of copper for electricity is reduced in a most striking manner by the presence of foreign matters, so that, in the construction of telegraphic apparatus, it is important that the purest attainable copper wire should be employed. Pure copper is scarcely inferior to silver (see p. 95) in its conducting power, and the conducting power of the native copper from Lake Superior, which is almost pure, stands to that of pure copper in the proportion of 93 to 100, whilst the Australian (Burra Burra) copper, also very pure, has a conducting power of 89, and the Spanish copper, which contains much arsenic, has a conducting power only one-seventh of that of pure copper, or in the proportion of 14 to 100.

The addition of a small proportion of phosphorus (about five parts in a thousand) to copper is found to harden it and somewhat to increase its tenacity; it is also said to render it less liable to corrosion when exposed to the action of seawater.

By adding arsenic to copper, in about the proportion of one to ten, a white somewhat malleable metal is obtained, which is not easily tarnished by air, and is much harder than copper. This compound, which is employed for clock dials and for thermometer and barometer scales, is made by heating five parts of copper clippings with two parts of white arsenic (arsenious acid) arranged in alternate layers and covered with common salt, in a covered earthen crucible.

TIN.

This metal is scarcely, if ever, found in the metallic state, but is extracted from the ore known as *tinstone*, which is an oxide of tin, or combination of the metal with oxygen. Cornwall has been noted for its tin mines from a very remote period; tinstone is also found in Australia, Bohemia, Saxony,

Malacca, and Banca, the *straits tin* obtained from the last-named localities being much valued on account of its purity. Siberia, Sweden, and North and South America also furnish tin ore, though in smaller quantity. Tinstone is found either as *stream tin ore* or *mine tin ore*. The former is also called *alluvial** *tin ore* from its occurrence in the mineral matter deposited by torrents in the valleys adjacent to the veins of mine tin ore, and is much purer than the latter, because it has been mechanically separated, by the action of the stream, from the foreign minerals which were associated with it in the vein. Occasionally, it is found in well-formed prismatic crystals which are perfectly pure oxide of tin. The mine tin ore occurs in veins traversing rocks of quartz, granite, or clay-slate, where it is associated with arsenical pyrites (see p. 224), copper pyrites, specular iron ore, and a remarkably heavy crystalline mineral called *wolfram* (tungstate of iron and manganese) which consists of tungstic acid (an oxide of the metal *tungsten*) combined with the oxides of iron and manganese. In order to obtain the tinstone in a sufficiently pure state for smelting, the ore is stamped to powder, washed, and calcined.

The processes which are put in operation in order to obtain marketable tin from the raw ore may be summed up under the following heads :

1. Mechanical preparation of the ore.
2. Calcining or roasting.
3. Washing the roasted ore.
4. Smelting.
5. Refining.

1. *Mechanical Preparation of Tin Ores.*—The mine tin ore, as it is raised from the mine, is roughly cleansed from earthy matters by washing it upon a grating under a stream of water. It is then picked over and broken with a mallet, the pieces of copper pyrites being placed aside to be smelted for

* *Alluvio* (Latin), an inundation.

that metal, and the iron and arsenical pyrites rejected. The tin ore is then crushed in the *stamp-chest* (C, Fig. 70) which is a wooden trough lined at the bottom with stamped ore, and provided with a number of massive wooden *stampers* (B) shod with blocks of cast iron weighing about $2\frac{1}{2}$ cwts. These are raised by cams fixed to an axle (A) which is made to revolve by water or steam power, so that each stamper may give about twenty blows in a minute, falling through a space of 8 or 10 inches. These heavy blows

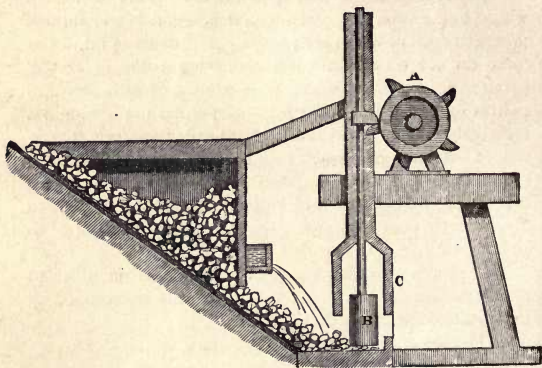


FIG. 70.—Section of Stamping-mill.

speedily reduce the ore to powder, and a stream of water, which constantly flows into the trough carries this powder through openings in three sides of it, which are closed with iron plates perforated with about 160 holes in the square inch, so that the larger fragments may not pass through. The holes in the iron plates are conical, having their narrower openings inside the trough, to prevent them from becoming choked. The water carries the powdered ore into a series of reservoirs, in which the ore settles down, whilst the water

flows away. Since the tinstone is much heavier than the other substances present in the ore (its specific gravity being



FIG. 71.—Rack for washing Ores.

6.5) the greater part of it is deposited in the first reservoir, the successive deposits becoming poorer as the stream flows on, and the sand, which has a specific gravity of about 2.7,

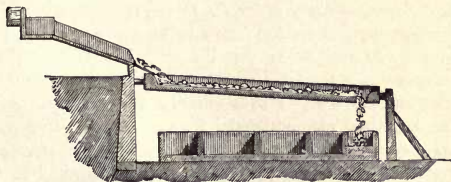


FIG. 72.—Section of Rack for washing Ores.

being in great measure carried away. Various mechanical contrivances are adopted for effecting a further purification of the *slimes* deposited in the reservoirs, in all of which

advantage is taken of the high specific gravity of the tinstone. The *rack* (Figs. 71, 72) which may serve as an illustration of these, is an inclined plane of wood with a shallow ledge (*g*), about 9 feet long, having one end 5 or 6 inches higher than the other. It is swung upon a pivot (*p*) at each end, so that it may be tipped and its contents emptied over the side. About 25 lbs. of the slimes are spread upon an inclined shelf (*h*), whence they are washed by a stream of water on to the inclined plane (*f*), when the sand and other earthy portions are carried away by the water, whilst the heavier tinstone, with some pyrites, &c., are left upon the plane; the deposit formed upon the higher portion of the incline is fit for the second process (calcining or roasting), but that formed on the lower part requires another washing. When a sufficient quantity of deposit has been collected on the table, the latter is tipped up sideways, and the upper and lower deposits allowed to fall into separate receptacles.

The *buddle* is a fixed inclined plane worked in a similar manner. The *tossing-tub*, *dolly* or *lieve*, is a tub in which powdered ore is stirred up with water and allowed to settle, the subsidence being hastened by striking the sides of the tub; the lower part of the sediment is of course the richest in the heavier tin ore.

2. *Calcining or Roasting the Tin Ore.*—The arsenical pyrites and copper pyrites are too heavy to be entirely removed by stamping and washing, so that the ore is next treated in the *burning-house*, where it is roasted in order to expel the arsenic and sulphur. This is effected in reverberatory furnaces, furnished with horizontal flues several hundred yards long, in which the white arsenic, formed by the arsenic in the pyrites and the oxygen of the air, is deposited in the solid state. From 6 to 8 cwts. of prepared ore is roasted at once, the temperature being very moderate at first, to avoid the fusion of the pyrites, and the ore being frequently raked over to expose fresh surfaces to the oxidising action of the air. The roasting occupies from 12 to 18 hours, and when

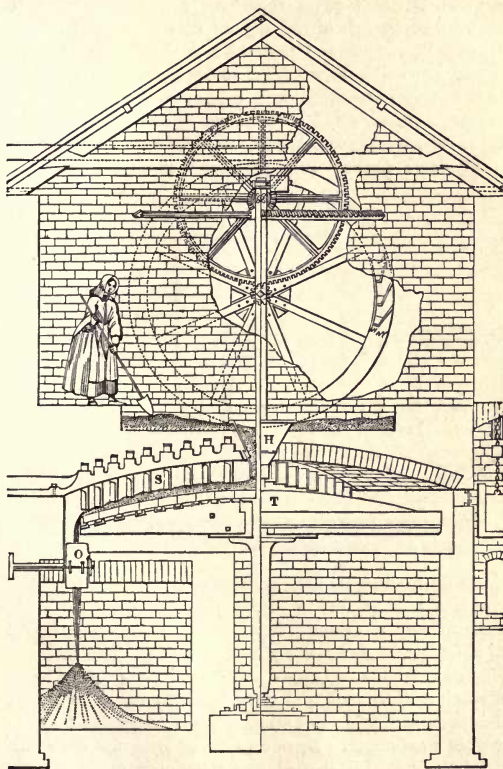


FIG. 73.—Brunton's Calciner.

it is completed, nearly the whole of the arsenic has been expelled in the form of vapour of arsenious acid (white arsenic), and much of the sulphur in the pyrites has combined with oxygen, and passed off as sulphurous acid gas ; a great portion of the sulphuret of copper in the copper pyrites has also combined with oxygen to form sulphate of copper, a change which is completed by allowing the roasted ore to remain exposed to the air, in a moist state, for some days.

Brunton's Calciner (Fig. 73), which is adopted in some

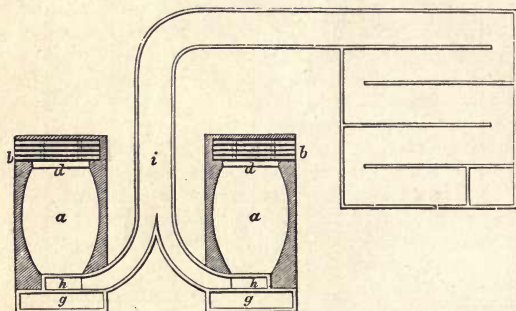


FIG. 74.—Saxon Furnaces for calcining Tin Ore. *a*, Hearth. *b*, Grate. *d*, Fire-bridge. *g*, Chimney. *h*, Flue. *i*, Channel for conveying the fumes into the condensing chambers.

works for roasting tin ores, consists of a circular table (T) of cast iron 8 or 10 feet in diameter, covered with fire-brick, made to revolve at three or four turns in an hour on the hearth of a reverberatory furnace with two grates. The tin ore is allowed to fall from a hopper (H) upon the centre of the table, where it is distributed and turned over by the *spider* (s), an iron frame with projecting arms, which is suspended from the arch of the furnace. By one of these arms the ore which has been gradually carried, by the rotatory motion, to the circumference, is delivered, in a roasted condition, through

an aperture (o) under the chimney. In this furnace the roasting operation is carried on without interruption.

In Saxony, the roasting is conducted by a wood fire in reverberatory furnaces (Fig. 74) connected with a flue above 100 feet long, in which the arsenious acid (white arsenic) produced by the oxidation of the arsenical pyrites is deposited. 12 cwt. are roasted in each furnace in 24 hours, yielding 5 or 6 cwt. of white arsenic.

Common salt is sometimes added to the tin ore previously to the roasting, when the chloride of sodium converts part of the arsenic and sulphur into chlorides which pass off in the form of vapour.

3. *Washing the Roasted Tin Ore.*—The next process consists in stirring the roasted ore with water, in a wooden tank, when the sulphate of copper is dissolved by the water, and is drawn off after the ore has settled down, the copper being recovered from the solution by leaving it in contact with iron, as in the case of the blue water of the Anglesea copper-mines (p. 248). By another washing upon the rack, or some similar arrangement, the lighter oxide of iron produced by the roasting of the pyrites is now removed, and the prepared tin ore or *black tin*, containing above 60 parts of tin in the hundred, is ready for smelting.

4. *Smelting the prepared Tin Ore.*—The furnaces (Fig. 75) generally employed in the *smelting-houses* of Cornwall are reverberatory furnaces, having a low arch, with air channels under the fire-bridge and hearth, to prevent injury from the high temperature, the draught being produced by a chimney 40 or 50 feet high. Coal is burnt upon the grate, the flame of which heats the material upon the hearth. About a ton of the prepared ore (oxide of tin) is mixed with one-fifth to one-eighth of its weight of ground anthracite coal, and with a little lime which is intended to flux or liquefy the small quantity of silica still mingled with the ore. Occasionally fluor spar is added with the same object. The mixture is damped with water, to prevent it from dusting, and thrown

upon the hearth of the reverberatory furnace. The doors are closed, but the temperature is kept low at first, for otherwise the oxide of tin would combine with the silica and the lime to form a glass or slag, causing a great loss of tin. The temperature is gradually raised for about five hours, by which time it should have reached its maximum. The charge is then well rabbled, and some culm having been thrown on the surface, the door is closed again, and heat applied for a further period of about three-quarters of an

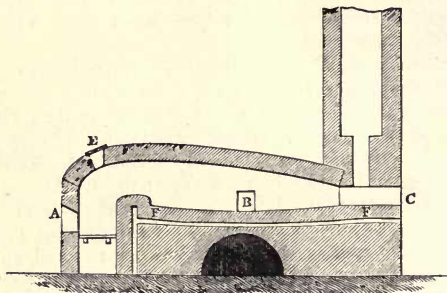


FIG. 75.—Furnace for smelting Tin Ores. A, Fire-door. B, Charging-door. C, Working-door. E, Door for moderating the draft whilst charging the furnace, lest the ore-dust be blown into the flue F, Air-channel under hearth.

hour, at the expiration of which the charge is again rabbled, and then left at rest for about fifteen minutes to regain the proper temperature for tapping. There are three principal products resulting from this smelting operation, viz. metal, glass, and slag. The 'glass,' which is the true slag, being perfectly fluid, is tapped out of the furnace with the metal. It is essentially a silicate of iron, but contains also any earthy matter from the ore, or added as flux, and a variable amount of tin as silicate. The glass, after it has been run from the furnace into the basin, separates into two layers.

The one called *bottle-slag*, resembling bottle-glass, is thrown away ; the other, a heavy black slag, is subsequently, when sufficient has accumulated, remelted. The *slag*, so called, is a scoriaceous pasty substance, consisting of anthracite, shots of tin, and 'glass.' The greater part of it remains behind in the hearth after the metal has been tapped ; it is removed through the end door, and subsequently stamped and washed to separate the tin.

5. *Refining the Metallic Tin.*—The ingots of tin, as obtained from the smelting-furnace, contain various impurities. Not only are particles of slag, and of the oxide of tin, entangled in the metal, but small quantities of iron, arsenic, copper, sulphur and tungsten are present in it, and must be removed in order to obtain marketable tin. This is effected by successive operations, which are known as *liquation* and *boiling*.

The process of liquation consists in melting out the tin and leaving the impurities behind. The ingots of tin are moderately heated upon the hearth of a reverberatory furnace similar to that employed for smelting the ore, when the bulk of the metal liquefies and is allowed to flow out of the furnace into a *refining-basin*, leaving a residue of impurities upon the hearth. Fresh ingots are introduced from time to time, until about five tons of tin have collected in the refining-basin, which is the case in about an hour after the commencement of the process.

The *boiling* consists in plunging into the tin contained in the refining basin, which is heated by a separate fire, stakes or logs of wet wood, which are held down under the metal by a lever-arm fixed above the refining-basin.

The heat of the melted metal causes a brisk evolution of steam, which throws the metal up, exposing a large surface to the action of the air, whereby the impurities become oxidised, and are removed from the surface in the form of dross. After about three hours the wood is taken out, and the

melted tin allowed to remain quiet for two hours, when the foreign matters which still remain dissolved in the tin gradually accumulate towards the bottom of the basin, leaving the upper part of the metal nearly pure. (Sometimes *tossing* is substituted for boiling, that is, the tin is well agitated by raising a ladleful of the metal to a considerable height, and pouring it into the bath.) The tin is then ladled out into moulds, either of granite or cast iron, and cast into ingots weighing about 3 cwt. each,* those cast from the upper or purer part of the metal being distinguished as *refined tin*, those from the middle layer as *common tin*, and those from the bottom being so impure that they must be again subjected to the refining process. The refined tin is very brittle at a temperature somewhat below its melting-point, so that when the ingots are heated and allowed to fall from a height, they break up into irregular prismatic fragments, which are called *grain tin*.† The refiner tests the purity of the tin by casting a small quantity in a stone ingot mould, when the refined tin remains bright and smooth after cooling, the common tin becomes partly frosted on the surface, and the impure tin is frosted all over.

The metallic residue left upon the hearth in the process of liquation allows tin of inferior purity to melt out when a stronger heat is applied ; this is run out into a small iron basin, allowed to rest, and the upper part of the melted metal ladled into moulds to be afterwards refined : a white brittle alloy, known as 'hardhead,' containing tin, iron, and other foreign metals, is found deposited at the bottom of the basin.

The refined tin will contain $99\frac{3}{4}$ parts of tin in the hundred, the common tin $98\frac{1}{2}$ parts, and the last portion,

* Banca tin is sold in blocks of 40 and 120 lbs. each ; Malacca tin in pyramids weighing about 1 lb. each.

† Only the best tin from specially pure ores is sold in the market as grain tin.

which requires a second refining, contains only 95 parts. The temperature at which tin is melted before casting is said to be of importance, its malleability being injured if it be cast at too low a temperature.

Reduction of Stream Tin Ore in the Blast-furnace or Blowing-house.—This operation is attended with greater consumption of fuel and loss of tin than that practised in England, but it is largely employed in the tin-works of Saxony. The blast-furnace (A, Fig. 76) is only ten feet high, cylindrical in shape, and surmounted by a conical hood divided into compartments for collecting the dust carried up by the blast, which is forced in by

a blast-pipe at *a*, near the bottom of the furnace. The sides and bottom of the furnace are built of granite, the bottom (D) being a single block hollowed out for the reception of the tin which flows out, together with the slag, into a basin of granite (B) lined with a coating of clay and charcoal. If the tin ore contains much oxide of iron,

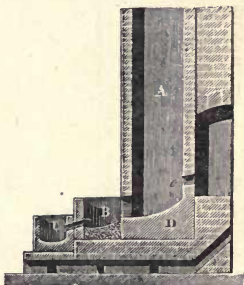


FIG. 76.—Blast-furnace or Blowing-house for smelting Tin Ores.

some quartz is employed as a flux; but if much silica is already contained in the ore, lime or finery cinder (p. 163) is employed to form a fusible slag.

The prepared tin ore and wood charcoal are constantly charged in at the top of the furnace, so as to keep it full, when the carbonic oxide, produced by the combination of the carbon with the oxygen of the air, abstracts the oxygen from the oxide of tin (see p. 132), and the metallic tin runs down to the bottom of the furnace, accompanied by a small quantity of slag formed by the fusion of the silica in the ore

with the ashes of the charcoal, and runs out into the *fore-hearth* (B), from which the slag is removed into a tank of water, the tin remaining liquid in the basin. When the latter is full of tin, it is discharged, through a tap-hole, into an iron basin (C), where it is further treated like the tin obtained by liquation (p. 259). The loss of tin in the blowing houses is twice or three times as great as in the smelting-houses, and is due to volatilisation of the metal.

At Altenberg, in Saxony, where the ores contain bismuth, they are treated, after roasting, with muriatic acid, for the extraction of that metal. The Saxony tin generally contains a little bismuth.

Treatment of Tin Ores containing Wolfram.—Since wolfram is even heavier than tinstone, its specific gravity being 7.5, and that of tinstone 6.5, the tin ore cannot be freed from wolfram by washing. Moreover, the compounds of tungstic acid obtained from wolfram have received, of late years, some important applications in the useful arts;

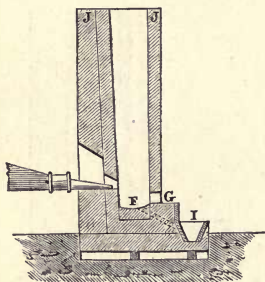


FIG. 77.—Blowing-house at Altenberg.
F, Crucible. G, Inclined plane for slag. I, Fore-hearth.

thus tungstate of soda is a most effective application for rendering muslin unflammable; tungstate of baryta is employed as a substitute for white lead. Accordingly, when the prepared tin ore contains any considerable proportion of wolfram, the quantity of this latter is ascertained by chemical analysis, and enough sulphate of soda (salt-cake from the alkali-works) is added to furnish a little more soda than is necessary to form tungstate of soda with the tungstic acid which is present. Some coal dust is added to the mixture, which is then

heated on the cast-iron hearth of a reverberatory furnace, when the carbon removes the oxygen from the sulphate of soda, leaving a combination of sulphur with sodium (sulphuret of sodium). A little air is then allowed access to the

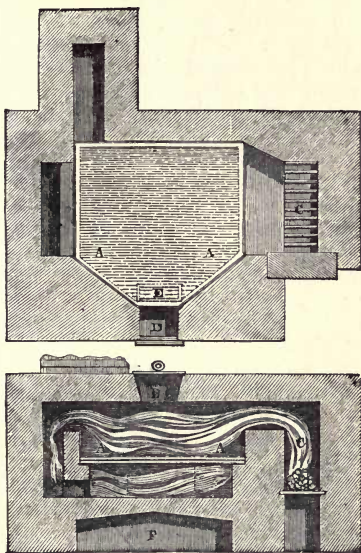


FIG. 78.—Reverberatory Furnace for manufacture of Tungstate of Soda from Tin Ores. B, Opening for introducing the charge. D, Working-door. E, Opening for discharging the contents of the hearth. F, Vault for receiving the finished charge.

heated mass, when its oxygen converts the sulphur into sulphurous acid gas, and the sodium into soda, which combines with the tungstic acid to form tungstate of soda. The furnace (Fig. 78) is so constructed that the flame from the

grate (C), after passing over the hearth (A) of the furnace, may return underneath it, so as to heat the charge uniformly. The mass is transferred from the furnace into tanks of water which dissolves the tungstate of soda, to be afterwards obtained in crystals by evaporating the solution. The oxides of iron and manganese derived from the wolfram are still contained in the tin ore, but they are so much lighter that they can easily be separated from it by washing, when the tin ore is ready to be smelted in the usual way.

Tin is remarkable for its property of *creaking* when bent; a bar of the metal, when bent to and fro, emitting a sound as if grains of sand were intermixed with the particles of metal. It has been noticed, in the general consideration of the properties of metals (p. 96), that tin is more easily melted than any other simple metal in common use, and that it is possessed of a high degree of malleability, which is turned to advantage in the manufacture of tin-foil, by rolling and hammering the metal into extremely thin leaves.

Tin is so little acted upon or corroded by air or by weak acids, that it is employed as a coating to protect the surfaces of other metals, such as iron, copper, and brass. Tin-plate is iron covered with a thin coating of tin, and since its manufacture is an important branch of English industry, an outline of it may be here given.

Manufacture of Tin-plate.—The sheet iron employed for the manufacture of the best tin-plate is refined with charcoal (p. 166), though iron refined with coke is sometimes employed, the tin-plate being distinguished accordingly as *charcoal-plate* and *coke-plate*. The last term, however, now usually refers to plate which has been made from puddled iron. A somewhat red-short (p. 188), iron from the Forest of Dean is extensively employed for the purpose, being possessed of great toughness when cold.

In order to obtain iron plates of the required thickness, the bars, $\frac{3}{8}$ ths of an inch thick, are cut into pieces 15 inches long and 6 inches wide. Each of these, having been heated

to redness, is rolled until its width is increased to 15 inches. It is then again heated, and rolled in the opposite direction until it is $5\frac{1}{2}$ feet long. The plate is next sprinkled with a little coal-dust to prevent it from sticking together, doubled, again heated, and passed between very smooth rollers until the original length of the doubled plate ($2\frac{3}{4}$ feet) has been increased to five feet. After being again doubled and heated, the four-fold plate is rolled out from 30 inches to 43 inches. It is then cut to the proper dimensions (not exceeding 18 inches by 13), the plates separated, and prepared for tinning. At this stage they are termed *black plates*.

The plates must be rendered perfectly clean and bright, for the slightest impurity would prevent the proper adhesion of the tin.

(1) The plates are bent so that they will stand on end, and arranged in a reverberatory furnace in order that they may be heated to redness.

(2) They are immersed in a mixture of four pounds of muriatic acid with three gallons of water for a few minutes, after which they are

(3) Again heated to redness, when the oxide of iron comes off the surface in scales.

(4) They are hammered straight and passed between rollers of cast iron hardened by chilling (p. 154).

(5) The plates are placed separately, on their edges, in sour bran-water, and occasionally turned, for ten or twelve hours.

(6) They are *pickled*, that is, immersed and stirred about for an hour, in a leaden trough containing diluted sulphuric acid, at a temperature of 90° or 100° F., until they are perfectly bright, the acid having dissolved off all the oxide. The sulphuric acid employed must be free from arsenic, which is apt to produce black spots upon the metal. This operation requires much care and attention on the part of the workmen.

(7) The plates are scoured with sand, under water, and are left under clean water (or sometimes under lime-water), which hinders rusting, until they are required.

(8) The brightened plates are rubbed with bran in order to dry them, and the drying is completed by leaving them for an hour in a cast-iron pot filled with melted tallow.

The process employed in some modern tin-plate works for preparing the plates to be tinned is much simpler than that just described.

(1) They are pickled in warm diluted sulphuric acid for about twenty minutes, which dissolves the black oxide of iron from the surface, the cleansing being completed by scouring them with sand and water.

(2) The plates are annealed by being heated to redness, for twelve hours, in an air-tight cast-iron box capable of containing 1,800 of them, placed in a reverberatory furnace; they are allowed to remain in the box till quite cold, when they are found to have acquired a deep purple colour, from a thin coating of oxide.

(3) They are *cold rolled* between very hard polished rollers, so as to toughen them.

(4) They are annealed as before for about six hours.

(5) The plates are again immersed for ten minutes in warm sulphuric acid, weaker than that employed in process 1, and finally scoured with wet sand.

Tinning.—(1) In order to tin them, they are transferred into another cast-iron pot containing melted tin covered with three or four inches of tallow, and heated over a fire to nearly the inflaming point of the grease. About 340 plates are immersed at once, and are allowed to remain in the tin for an hour and a half, or even longer, according to their thickness, after which the superfluous tin is allowed to drain off by placing them upon an iron grating.

In this operation of tinning, an alloy of iron with tin is formed upon the surface of the plate, and a firm adhesion of the tin coating is thus secured.

(2) The next operation, *washing*, is intended to equalise the coating of tin, and to give the plate a smooth bright

surface. For this purpose, an iron pot with two compartments is employed, both of which are filled with melted grain tin, of high purity, one being designed to receive the superfluous tin from the plate, and the other to give a final coating of pure tin. The plates are immersed in the first compartment, and when sufficiently heated, they are lifted out by tongs, and brushed on each side with a hempen brush, they are then dipped for a moment into the pure tin in the second compartment, which removes the marks of the brush, and afterwards plunged into melted tallow, where they are left for a certain time, and at a particular temperature, in an upright position, when the superfluous tin drains down to the lower edge and forms a rim or *list* which is removed in the next operation. After a certain number of plates have been dipped into the first division of the washing pot, and the tin has become rather impure, about 3 cwt. of it is ladled into the first tinning pot, and replaced by an equal weight of grain tin.

(3) The plates having been placed in another vessel, upon an iron grating, to cool, their lower edges are dipped into the *list-pot*, which contains a layer of melted tin about a quarter of an inch deep; as soon as the list is melted, the plate is lifted and smartly struck with a stick, when the rim of tin drops off.

After the grease has been removed by rubbing the plates with bran, they are ready to be packed in the boxes, which contain 100, 200, or 225, according to the description of plate. About $\frac{1}{2}$ oz. of tin is sufficient to cover a plate of iron measuring 14 inches by 10.

Tin-plate is very durable as long as the coating of tin is perfect, but if any portion of the iron surface beneath should be exposed, it is corroded by rusting even more rapidly than untinned iron plate, because the iron and tin, in presence of the film of moisture containing carbonic acid which is deposited by the atmosphere upon the surface of the plate, form a galvanic pair of which the iron is the metal attacked by the water and acid, so that the plate is speedily eaten

through. In the manufacture of the best kind of tin-plate, technically termed *doubles*, and popularly, *block-tin*, a thicker coating of tin is given, and its perfect union and consolidation with the iron plate is ensured by going over the entire surface with a polished hammer upon a polished anvil, an expenditure of manual labour which of course greatly increases the cost of the article.

Tin-plate moiréed.—The beautiful variegated appearance known as the *moiré métallique* is produced by wiping the surface of tin-plate with tow or sponge dipped in a warm mixture of diluted nitric acid with hydrochloric acid, or with common salt, or sal-ammoniac, and well washing with water. The acid liquid dissolves away the smooth surface of the tin and discloses the crystalline structure beneath, the variegated appearance being apparently caused by the reflection of the light in a myriad different directions by the minute faces of the crystals. The *moiré* may be greatly diversified by heating the plate before applying the acid, and cooling it irregularly by sprinkling water over it, or by directing the blow-pipe flame over its surface before wetting it with the acid. The surface is afterwards covered with a transparent coloured varnish.

Tinning of Copper.—Copper saucepans, stewpans, &c., should always be coated with tin, since most kinds of food are capable of dissolving a little copper, and the poisonous effect of the compounds of this metal becomes perceptible even when very small quantities are present. Fortunately it is much easier to tin copper than iron. The copper surface having been smoothed by rubbing it with a fine sandstone, is made pretty hot, and rubbed over with powdered sal-ammoniac, which has the property of removing the oxide from the surface, and leaving the copper perfectly bright. A little tin is now placed upon the copper, together with some powdered rosin, the latter being intended to form a varnish when it melts upon the surface of copper, which it protects from being oxidised by the air. The copper plate is again heated, and

when the tin melts, it is spread with tow over the surface, to which it firmly attaches itself. 200 grains of tin are commonly employed to cover a square foot of copper. The tin employed for this purpose is sometimes alloyed with lead.

ALLOYS OF TIN AND COPPER.

The term *alloy* (from the French *allier*, to blend or unite) is applied by metallurgists to any material which is produced by melting two or more metals together. The properties of the metals which are thus alloyed with each other generally undergo a much greater alteration than can be accounted for, if it be supposed that the alloy is a mere mechanical mixture of its constituent metals, though the proportions in which metals are capable of being alloyed with each other are not so fixed and definite as is the case with substances entering into chemical combination with each other. Since, however, chemists are able to produce, in some cases, very definite chemical combinations of one metal with another, it seems probable that the alloys consist usually of such definite chemical compounds, dissolved in, or mingled with, an excess of one of the constituents over and above the quantity which is actually required to take part in the formation of a chemical compound.

The alloys of tin and copper may be cited in illustration. When two parts of copper are melted together with one part of tin, the mass, after cooling, is found to possess properties very different from those of either of the metals, being very hard, almost as brittle as glass, and having a white silvery fracture. No merely mechanical mixture of two soft malleable metals could produce one which would be hard and brittle, so that the conclusion appears inevitable that a chemical combination has taken place and that a new material has been produced. The tenacity of this alloy is only about one-fifth that of tin, and one-fiftieth that of copper, and it is increased by the addition of either of these metals, which appears to indicate that a further quantity of either of

them merely becomes mixed with the alloy. When this alloy is melted with more copper, it is entirely dissolved as long as the metal is liquid, but during the cooling, the alloy tends to separate into two parts, one containing more copper, which solidifies first, and the other much richer in tin. This result would indicate that the original alloy had not formed a true chemical combination with the excess of copper, but had been merely dissolved by it.

The alloy of two parts of copper and one part of tin forms the basis of the *speculum metal* of which the mirrors of reflecting telescopes and other optical instruments are made, arsenic being sometimes added in the proportion of about one-tenth of the weight of the tin, in order to harden the alloy and render it susceptible of a finer polish. A little zinc is often added with the same object. In order to make the alloy, the copper and tin are melted in separate crucibles, and stirred together with a piece of wood. Care is necessary in employing the proper proportions of the metals, for an excess of tin imparts a bluish tinge to the alloy.

Bell-metal is an alloy of copper and tin, the proportions of which are varied according to the size of the bells. Large bells are cast with an alloy of four parts of copper and one part of tin, which is also the composition of the alloy employed for cymbals and gongs. The metal, when first cast, is exceedingly brittle, but it becomes somewhat malleable after being heated to redness and quenched in water. To give it the elasticity which is necessary in order that it may emit a full clear sound, the bell is again heated and allowed to cool slowly. There is an art in the manufacture of a good gong which appears to be possessed by the Chinese alone, and has not yet been successfully imitated in this country.

Gun-metal is composed of $90\frac{1}{2}$ parts of copper and $9\frac{1}{2}$ parts of tin ; it is harder and more fusible than copper, and presents great resistance to any strain tending to force its particles asunder. In consequence of the great difference in the specific gravities of the metals, it would scarcely be

possible to mix them thoroughly if they were simply introduced together into the reverberatory furnace in which the alloy is prepared. It is customary, therefore, to melt the tin first with twice its weight of copper, so as to obtain *hard metal*, which is then added to the proper proportion of copper melted on the hearth of the reverberatory furnace, care being taken to exclude the oxygen of the air from the hearth, and to mix the metals thoroughly by stirring them with a wooden pole. The formation of the alloy is facilitated by the addition of some old gun-metal. A little more than the necessary proportion of tin is usually added, in order to allow for the unavoidable conversion of a portion of that metal into oxide by the oxygen of the air. When the metals are thoroughly mixed, the oxide is skimmed from the surface, and the gun-metal is tapped into moulds made of loam, the stirring being continued while it is running, to counteract the tendency of the alloy to separate into two parts, as above alluded to. For the same reason, the alloy is run into the mould at a temperature as little as possible above its point of solidification, so that it may not long remain liquid in the mould. In spite of these precautions, a partial separation of the metals always takes place, so that the upper portion of the casting contains more tin than the lower. On this account, it is usual to cast guns with their muzzles upwards, in moulds which are prolonged, in the form of an inverted truncated cone, to about three feet beyond the required casting; the excess of metal, amounting to about twice the weight of the gun, forms what is called a *dead-head*, the weight of which tends to prevent the separation of the metals in the lower part of the casting. This dead-head exhibits a kind of ebullition during the solidification, and a considerable separation, at its upper part, of an alloy containing more than twice as much tin as gun-metal. The dead-heads are cut off before the guns are turned and bored, and are fused in the furnace where the gun-metal is prepared. When the white alloy, rich in tin, separates to any great extent in the lower part of the casting, it produces flaws, or

tin spots, and the gun is rejected. Comparatively few bronze guns are now cast, since wrought iron and steel have been largely employed for the manufacture of ordnance, being much lighter and stronger.

Bronze has essentially much the same composition as gun-metal, but great variations are made in the proportions of copper and tin, in order to adapt it for special purposes, small quantities of lead and zinc being also occasionally added.

The bronze coinage of this country contains 95 parts of copper, 4 parts of tin, and 1 part of zinc.

The effect of tin in hardening copper was well known to the ancients, who made swords, scythes, nails, &c., of bronze, before the art of working iron and steel had been acquired. The mode of tempering such bronze weapons was just the reverse of that practised with steel, the bronze being rendered soft and somewhat malleable when heated and quenched in water, but hardened again by being heated and cooled slowly.

Tin having a much greater attraction for oxygen than copper has, a considerable proportion of the former metal is lost, as oxide, when bronze is remelted.

The bronze-founder employs a reverberatory furnace so arranged as to prevent access of air to the metal. By the unskilful remelting of old bronze guns, the tin has been reduced to less than half the original proportion.

Britannia-metal.—Tin constitutes the chief part of this and some similar alloys which are employed for the manufacture of spoons, tea-pots, &c., being hardened for such purposes by the addition of antimony and copper. Lead and bismuth are also sometimes added.

The following modifications of bronze are employed for particular purposes:—

	Wheel-boxes or sockets	Stop-cocks and pump-valves	Nails for ships' sheathing
Copper . . .	80	88	87
Tin . . .	18	10	9
Zinc . . .	2	2	4
	<hr/> 100	<hr/> 100	<hr/> 100

ZINC.

Metallic zinc is not met with in nature, and though its combinations with other substances are abundant in certain localities, they are by no means universally diffused over the earth's surface. England is not particularly rich in ores of zinc, and the extraction of the metal is carried out in this country to a very limited extent, most of the zinc required in the arts being imported from Silesia, Belgium and Poland.

The ores of zinc from which the metal is extracted are enumerated in the following table :—

Ores of Zinc.

	Composition	Zinc in 100 parts of pure ore
Blende . . .	Zinc, Sulphur	67
Red zinc ore . . .	Zinc, Oxygen	80
Calamine . . .	{ Zinc, Oxygen, Carbonic Acid }	52

Blende derives its name from the German *blenden*, to dazzle, in allusion to its lustre. It usually occurs in black shining crystals which owe their colour to the presence of sulphuret of iron, since the pure compound of zinc with sulphur is white. Blende is also met with of a brown or yellow colour. Black blende is sometimes regarded as a definite compound of sulphuret of zinc and sulphuret of iron, containing 52 parts of zinc in the hundred. The chemical name of blende is sulphide, or sulphuret, of zinc, and the miners often call it *Black Jack*. It is found running in veins through limestone or sandstone, and is commonly associated with galena (sulphuret of lead) and with iron and copper pyrites. Blende occurs in Cornwall, Devonshire, Cumberland, Derbyshire, Ireland, Wales, and the Isle of Man; also at Freiberg, Aix-la-Chapelle, and in North America. It sometimes contains a considerable proportion of cadmium.

Red Zinc Ore is the oxide of zinc, which would be white in

the pure state, but is coloured in this ore by the oxides of iron and manganese. It sometimes forms red translucent prismatic crystals, and is found chiefly in New Jersey, in the United States, where it is first smelted for zinc, and afterwards for white pig-iron.

Calamine appears to be so called in allusion to the columnar structure of some specimens of the ore, which gives them some resemblance to a bundle of reeds (*calamus, a reed*). It is a compound of the oxide of zinc with carbonic acid, which would be white if pure, but is usually of a buff or brown colour, due to the presence of oxide of iron, which is objectionable, because it corrodes the clay vessels employed in smelting the ore. Calamine occurs in veins, commonly traversing limestone rocks, and is associated with blende, galena, and *electric calamine*, which resembles calamine in appearance, but becomes electric when heated. The electric calamine is a compound of oxide of zinc, silica, and water (hydrated silicate of zinc), and though it is pretty abundant and rich, it can scarcely be regarded as an ore of zinc, for it does not yield its zinc in the ordinary process for extracting the metal. Calamine is found in Flintshire, the Mendip Hills in Somersetshire, Alston Moor, in Cumberland, at Lead Hills in Scotland, at Aix-la-Chapelle, at Tarnowitz in Silesia, in the north-west of Spain, and in many other places. It sometimes contains more than two parts of cadmium in the hundred. In Spain, the carbonate of zinc is found in combination with the *hydrated oxide* of zinc, so that the ore contains as much as 57 parts of zinc in the hundred. Beds of calamine are reported to have been recently found in Sardinia.

The chief English zinc-works are situated in Birmingham and Bristol, where the ores from the Mendip Hills and Flintshire are smelted; in Sheffield, where the ore is procured from Alston Moor; and at Swansea, Wigan, Llanelly, and Wrexham.

In order to extract zinc from its ores, advantage is taken

of the comparative facility with which the metal is converted into vapour, since it boils and distils freely at a temperature estimated at about 1900° F., a bright red heat, somewhat below the melting point of copper. The ores are calcined so as to obtain the zinc in the form of oxide, which is then mixed with carbon and distilled, when the oxygen passes off in combination with the carbon as carbonic oxide gas, and the zinc is given off in vapour which is condensed again. The mode in which the operation is carried out differs in different works, but the principle of the process is always the same.

Calamine is the principal ore treated in this country, and is sometimes smelted without previous calcination, because the carbonic acid which is combined with the oxide of zinc can be driven off in the smelting process itself; but the calcination or roasting of blende is indispensable, to enable the oxygen of the air to convert the zinc into oxide, and to carry off, in the form of sulphurous acid gas, the sulphur previously in combination with the metal. Care is taken to pick out as much of the galena (sulphuret of lead) as possible, because the oxide of lead which would be formed from it would combine with the silica of the earthen crucibles employed in the smelting process, and would seriously corrode them. The blende is also stamped to powder and washed to free it from earthy matters before calcining.

The ore having been broken into fragments of the size of a nut, the calcination or roasting is effected, as usual, by the flame of a coal fire, in a reverberatory furnace about ten feet long and eight wide, about a ton of ore being spread upon the hearth, and occasionally raked over. The roasting is completed in 10 or 12 hours.

Blende is sometimes subjected to a preliminary roasting in heaps (p. 144), to expel a part of the sulphur before introducing it into the reverberatory furnace.

The roasting-furnace has very commonly two hearths, one above the other, so that the flame, having traversed the sur-

face of one, must pass across the surface of the second hearth before reaching the chimney. The ore is roasted on the upper hearth, which has the lower temperature, for 12 hours, and afterwards, for about the same period, upon the lower hearth. At Moriston, near Swansea, the roasting furnace is about 36 feet long and 9 feet wide, the hearth being divided into three steps, that nearest the fire-bridge being eight inches lower than that near the chimney, and the middle one of intermediate height. 12 cwt. of blende are

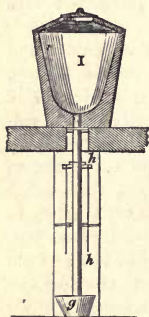


FIG. 79.—Crucible for extraction of Zinc by the English method.

roasted for eight hours on the coolest hearth near the chimney, then for eight hours on the middle hearth, and finally, for a similar period, upon the hottest hearth near the fire-bridge, fresh charges having been introduced at the other end.

In order to save the clay vessels in which the distillation is effected, those ores which contain much oxide of iron and lime are mixed with others containing clay, which is attacked by those substances, instead of the material of the distilling vessels.

The old English method of effecting the distillation is now almost obsolete, either the Belgian or Silesian process having been adopted in the larger zinc works, but the process is sufficiently interesting to merit a short description.

Old English Method of Extracting Zinc.—The roasted ore is distilled with coke in crucibles (Fig. 79) made of Belgian fire-clay (nearly pure silicate of alumina), furnished at the bottom with iron pipes in which the zinc is condensed. The crucibles are made, generally on the premises, of a mixture of equal parts of fire-clay and old crucibles ground to powder, and each crucible will commonly last about four months, in which period it distils two tons of zinc. The

crucibles are 4 feet high and $2\frac{1}{2}$ feet wide, and the iron pipes which pass through the bottom of each crucible are seven inches wide. These pipes are made in two pieces, the shorter length being cemented into the bottom of the crucible, and the other (*h*), about eight feet long, being made to fit on to it when necessary. Six of these crucibles are

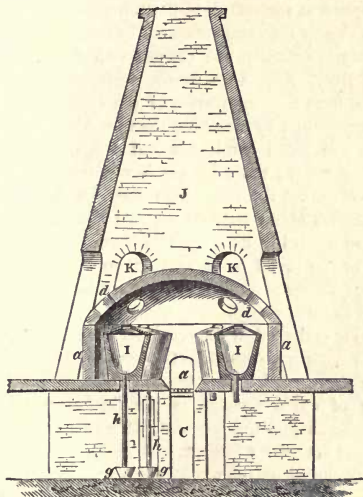


FIG. 80.—Extraction of Zinc by the English method.

usually set in one furnace (Fig. 80), which resembles that used in a glass-house, having a long grate (*a*) running through the centre, on which a brisk coal fire is maintained, the flame from which circulates round the crucibles, and issues through the six apertures (*d*) in the dome (*a*). The external cone (*j*) serves as a chimney to carry off the smoke, and as a jacket to retain the heat.

The pipe at the bottom having been loosely plugged with a lump of coke to prevent the charge from passing into it, each crucible is charged, through the opening (*d*) above it in the dome, with 4 or 5 cwt. of the calcined ore and 2 or 3 cwt. of broken coke or anthracite. The covers are then cemented on and the fire applied. In a short time, a blue flame appears at the mouth of the tube, beneath the furnace, which is due to the combustion of the carbonic oxide formed by the combination of the carbon with the oxygen of the oxide of zinc. After a few hours, the flame becomes greenish white, from the combustion of the vapour of zinc; it is then extinguished by attaching the longer piece of iron pipe, in which the metal condenses and drops into a vessel (*g*) containing water; the tube (*h*) is removed occasionally in order to clear out the zinc which obstructs it. The distillation occupies about sixty hours, and the ore yields rather more than a third of its weight of zinc, a considerable proportion of the metal being still left in the form of electric calamine (silicate of zinc) which cannot be made to yield up its metal in this process.

The zinc collected in the receivers is, of course, not in the form of a compact mass of metal, because it distilled over in separate drops, and it is intermingled with a considerable quantity of oxide formed as the heated metal dropped through the air into the receiver. It is therefore remelted in an iron pot set over a furnace, and well skimmed, the dross being introduced into the crucibles with a fresh charge. The zinc is cast into flat cakes or ingots, and sent into commerce under the name of *spelter*.*

When cadmium is present in the zinc-ore, it passes over in vapour before the zinc, for cadmium boils at 1580° F., and its presence is indicated by the brown fumes of oxide of cadmium which rise from the flame (*brown blaze*). The metal which then distils over consists of zinc containing a

* The name *spelter* is also given to an alloy of equal weights of zinc and copper employed for soldering brass.

large proportion of cadmium, and is collected separately, the cadmium being employed in making a particular kind of fusible alloy used by dentists, whilst its compound with sulphur forms a yellow paint known as *cadmia*, and other of its combinations are useful to the photographer.

Extraction of Zinc from its Ores by the Belgian Process.—

This process is now that most employed in England. The distillation of the mixture of ore and coal is effected in cylinders which are made usually in the zinc-works themselves, from a mixture of raw and hard-baked fire-clay. They are about 8 inches wide and 3 feet long, and are closed at one end. A large number of these cylinders, varying from 40 to 80, are arranged in tiers (*a b c d* Fig. 81), so that they may be heated by the same fire (F), the mouths of the cylinders being placed a little lower than the closed ends. Two, or even four, furnaces are built in the same block, each with two flues running into a common chimney, which is divided into four compartments, each having a separate damper. The furnaces near Swansea are 11 feet wide, $9\frac{1}{2}$ feet high, and 4 feet deep, containing 78 cylinders.

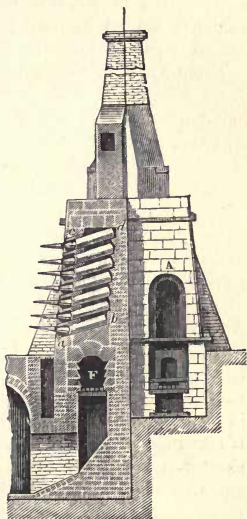


FIG. 81.—Belgian Furnace for distilling Zinc from its Ores.

Coal is burnt on the grate, and its flame penetrates into the furnace through four openings beneath the lowest tier of

cylinders. Of course the lower cylinders are thus raised to a higher temperature than the upper rows, so that the former are charged with about 24 lbs. of the mixture of ore and coal, and the latter with 16 lbs. Before the cylinders are arranged in the furnace, the interior of the latter is gradually brought to a high temperature, the front wall being temporarily closed in for that purpose. After four days, the cylinders are set in their places, having been previously heated to redness in a separate furnace.

The mixture of ore and coal, slightly moistened to prevent dusting, is then introduced into each cylinder, to the mouth of which is cemented a conical tube of fire-clay, or of cast iron, about ten inches long. In a short time, the flame of carbonic oxide gas, formed by the carbon with the oxygen from the oxide of zinc, is perceived at the mouth of the tube, and soon afterwards acquires the brilliant greenish-white appearance which indicates that the zinc is beginning to distil over. In order to condense the vapours of zinc, a conical pipe of sheet iron, the smaller opening of which is only two-thirds of an inch in diameter, is now attached. After the lapse of two hours, this is removed and cleared out, the mixture of zinc and oxide of zinc which it contains being worked again with the next charge. The bulk of the zinc has condensed in the cast-iron cones, which are placed in such a position that the melted zinc remains in them, and is raked out by the workmen into a large iron ladle. The sheet-iron cones are again attached, to exclude the air, and after another interval of two hours the condensed zinc is raked out as before. Twelve hours are required to distil the zinc from a single charge of ore, after which a fresh charge is at once introduced, so that the furnaces are kept in continual operation for two months, when they are stopped for repairs. 100 lbs. of the ore furnish, on an average, 31 lbs. of zinc, but a considerable proportion of zinc still remains, combined with silica, in the residue, since the extremely high temperature required to extract it would soften the clay

cylinders and cause them to collapse. On account of the high price of coal, clay, and labour, ores containing less than 40 parts of zinc in the hundred cannot be worked in Belgium.

The zinc collected in the iron ladle is skimmed from dross, and cast into ingots weighing from 70 to 80 lbs. each.

At the Vieille-Montagne works, near Aix-la-Chapelle, the ore, a part of which is imported from Sweden, consists of two kinds of calamine, distinguished as *white ore* and *red ore*, the latter containing more iron and less zinc than the former. After being freed from clay by washing, the ore is calcined, either in reverberatory furnaces, or in kilns resembling lime-kilns, in which coal is employed as fuel; it loses about one-fourth of its weight. The ore is then ground to a fine powder, sifted, mixed with half its weight of coal-dust, and distilled in the furnace described above, the white ore, which is richer in zinc, being introduced into the lower rows of cylinders.

Silesian Process for Extracting Zinc from its Ores.—In Silesia, ores which contain only 18 or 20 parts of zinc in the hundred are worked with profit. The calcined calamine is distilled with coal or coke in large *muffles*, or arched ovens (4, Fig. 82), 3 or 4 feet long, by 8 inches wide and 18 or 20 inches high. They are made of fire-clay mixed with ground pots, like the cylinders employed at the Vieille-Montagne, but their flat bottoms being well supported throughout their whole length, the muffles will sustain a higher temperature than the cylinders without bending, so that the distillation of the zinc is much more completely effected. The calamine is calcined in reverberatory furnaces, which are sometimes heated by the waste heat of the smelting-furnace. It is broken up into grains about as large as a pea, mixed with about an equal bulk of broken coke or fine cinders, and introduced into the muffle through an opening which is afterwards closed with a fire-clay stopper. The muffles are provided with rectangular earthen tubes (*p*), prolonged by a

cast-iron cone and a sheet-iron tube, for the passage and condensation of the vapour of zinc, an opening (*q*) being provided through which the tubes may be cleared from obstruction. Twenty of these muffles are arranged in each furnace, so that the flame may pass well round the top and sides of them, and the firing is very gradual, to avoid all risk of

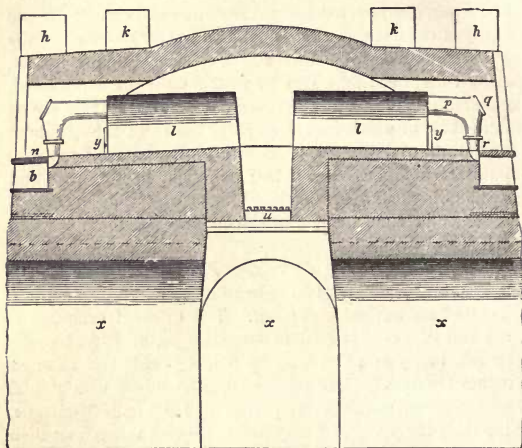


FIG. 82.—Silesian Zinc-furnace ; section. *l*, Arched retorts. *y*, Door for removing the exhausted ore. *q*, Door for charging the retorts. *p*, *r*, Tube for condensing the zinc. *b*, Receptacle for condensed zinc. *k*, *k*, Flues. *u*, Fire-grate. *x*, Vaults.

splitting them. The condensed zinc drops from the earthen tube through an opening corresponding to it, in the fore-hearth of the furnace, and is collected in a small chamber (*b*) beneath. The muffles are charged afresh every 24 hours, and last about 27 days. The residue left in the muffle does not retain more than $\frac{1}{50}$ th of its weight of zinc. The zinc

is remelted in iron pots lined with clay, since, if the melted metal be in direct contact with the iron vessel, the latter is corroded, and the zinc becomes contaminated with iron, which injures its quality.

To smelt 1 cwt. of zinc from the ore, the coal consumed

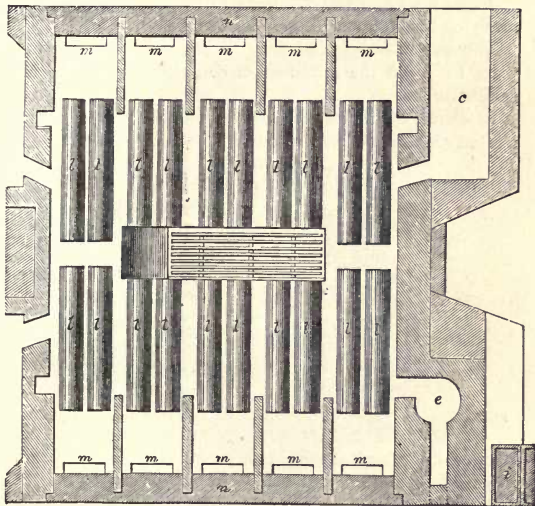


FIG. 83.—Silesian Zinc-furnace; plan. *z*, Arched retorts. *c*, Space for heating the retorts before setting them in the furnace. *e*, Pot for remelting the distilled zinc. *i*, Iron ingot-moulds for casting the zinc.

is, in Belgium, from 6 to 8 cwts., in Silesia from 10 to 15 cwts., and in England 25 cwts.

In the Belgian-Silesian furnaces, as they are called (Fig. 85), the flame is made to pass completely round the muffles (*a*, *b*), and to descend beneath the furnace to the chimney. Thirty-two muffles are commonly arranged in each furnace.

The condensing-tubes (*g*) have a depression at the under side, in which any lead which distils over is deposited, and the zinc runs off in a purer condition, and are sometimes provided with tapping-holes through which the zinc may be run off into ingot-moulds. At Stolberg, a furnace of this construction has 60 muffles arranged in two tiers, each muffle receiving 70 or 80 lbs. of ore.

The only substantial advance in zinc smelting in recent years has been the introduction of gas-firing and the regenerative system.

In some cases it has been found economical to charge the roasted ore into a cupola or small blast-furnace, in alternate

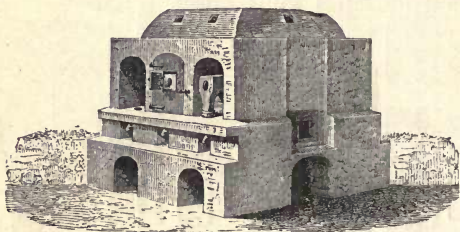


FIG. 84.—Silesian Furnace for distilling Zinc from its Ores. *z*, Condensing tube.
z, Openings beneath the hearth for receiving the condensed zinc.

layers with fuel, when the zinc is given off in vapour, which is speedily converted into oxide by the oxygen of the air, and, being collected in condensing-flues, is distilled in the usual way with coal for the production of metallic zinc. The advantages of this process consist in the greater yield of zinc in the distillation, and the absence of the oxides of lead and iron, which do so much damage to the clay vessels.

At Bleiberg, the *zinc-dust*, or mixture of finely-divided zinc with oxide of zinc, which is first collected during the distillation of the ores, is melted in fire-clay tubes set upright in a furnace; clay pistons attached to iron rods are

thrust into the tubes, the pressure causing the finely-divided zinc to run together, so that it may be tapped out from the bottom of the tubes. The metal so obtained is very impure, containing much arsenic and cadmium, which always pass over with the first portions of distilled zinc.

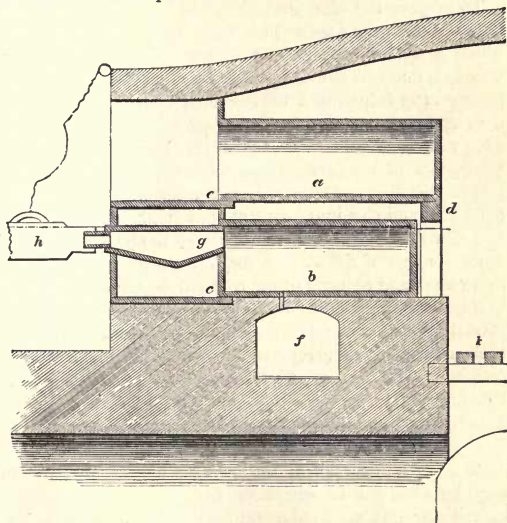


FIG. 85.—Belgian-Silesian Zinc-furnace. *a b*, Arched retorts. *c d*, Their supports. *f*, Flue. *g*, Condensing tube. *h*, Receiver suspended by iron-wire. *k*, Fire-place.

Rolling of Zinc into Sheets.—Before the year 1812, zinc was used almost exclusively for the manufacture of brass, since it is brittle both at the ordinary temperature and at high temperatures, but it was then discovered that a temperature between 200° and 250° F. rendered it malleable and capable of being rolled into thin sheets. For this

purpose, however, it is necessary that the zinc should not contain iron or lead, the former of which it acquires when remelted in iron pots, while the lead is carried over in the distillation of the zinc, in consequence of the presence of galena (sulphuret of lead) in the ore.

To prepare the zinc for rolling, the ingots of spelter are remelted upon the fire-clay hearth of a reverberatory furnace, which is made to slope down to a deep cavity or *sump* at one end, into which the melted zinc flows. Lead (specific gravity 11.4) being much heavier than zinc (specific gravity 6.9), and the two metals having very little disposition to alloy with each other, the bulk of the lead settles down to the bottom of the cavity, so that the upper portion of the melted metal is comparatively free from lead. It is then ladled out and cast into ingots of convenient size for rolling, and after these have been again heated to about 250° F. by the waste heat of the furnace, they are passed between cast-iron rollers and reduced to the required degree of thinness.

After being rolled, the zinc always retains a great deal of its malleability, even after cooling, whereas cast zinc may be broken by laying it across an anvil and striking it with a hammer. The fracture of an ingot of zinc presents a very beautiful crystalline appearance and great lustre, and if it contain iron, minute grey spots are visible on the bright faces of the crystals. At temperatures above 400° F. zinc is even more brittle than at the ordinary temperature, and it may be obtained in extremely fine powder by pouring melted zinc into an iron mortar, and well stamping it with the pestle as it solidifies. Powdered zinc has been employed as a metallic paint for protecting iron from rust.

Since the discovery of the malleability of zinc at very high temperatures, this metal has been extensively employed for gutters, rain-pipes, cisterns, baths, chimney-pots and roofing, purposes for which it is eminently fitted by its lightness and by its resistance to the action of air and moisture; for although a bright surface of zinc soon tarnishes, from the

formation of a film of oxide of zinc upon it, this film serves to protect the metal beneath from any further corrosion.

An objection to the use of zinc for roofing is the great combustibility of the metal, for, at a red heat, it takes fire and blazes fiercely, producing light white flakes of oxide of zinc, which is used as a paint, under the name of *zinc-white*.

In casting zinc, it is important to avoid employing too high a temperature, not only because zinc may be lost in vapour, but because *burned zinc* is produced; that is, the zinc becomes very much harder, and difficult to cut with a file or chisel, probably because it has dissolved some oxide of zinc.

Galvanised Iron is sheet iron coated with zinc, and is made in a similar way to tin plate, by thoroughly cleansing the surface of the iron, and immersing it in melted zinc which is kept covered with powdered sal-ammoniac (muriate or hydrochlorate of ammonia), this salt having the property of dissolving the oxide of zinc from the surface of the bath. The zinc probably alloys with the iron plate to a slight depth. A small quantity of iron is dissolved by the melted zinc, and a very brittle alloy is deposited in pasty masses at the bottom of the zinc-pot, whence it is removed occasionally with a perforated ladle. This alloy has been found to contain 6 parts of iron and 94 parts of zinc. In some large galvanised iron works, where much of this alloy is obtained, the zinc is recovered by distilling the alloy in clay retorts like those employed in gas-works. A process recently devised for treating this alloy consists in melting it in iron pots, and cooling it slowly, from the bottom upwards, when an alloy of $9\frac{1}{2}$ parts of iron with $90\frac{1}{2}$ parts of zinc is deposited in crystals, which are removed by a perforated ladle, and the liquid zinc is left much purer. The crystallised alloy is distilled, to recover the zinc. The zinc coating adheres more firmly if the iron is previously tinned.

Iron articles are also coated with zinc by connecting them with wires attached to the negative pole of a weak gal-

vanic battery, and immersing them in a solution of sulphate of zinc; this is decomposed by the galvanic current, the zinc being deposited upon the surface of the iron, which is thence said to be *galvanised*. When any portion of the iron surface is exposed in consequence of the abrasion of the zinc, the adjacent portion of the latter metal will protect the iron from corrosion, because the two metals form a galvanic pair of which the zinc only is attacked by the moisture and carbonic acid of the air. It will be remembered that, in the case of tin plate, it is the iron which is the metal attacked.

Galvanised iron is ill-adapted for situations where it is much exposed to the acid vapours sent into the air by some factories, or to the sulphuric acid found among the products of combustion of coal and gas, because zinc is among the metals most easily attacked by the acids. It is often called *corrugated iron*, from the practice of ridging and furrowing the plates in order to strengthen them for building purposes. Vessels of galvanised iron are not well fitted for cooking utensils, since many articles of food are liable to become contaminated with zinc to a hurtful extent.

The low temperature at which zinc is melted (770° F.), and its perfectly liquid condition, recommend it for casting statues, since it runs easily into the very finest lines of the mould. It is only a sixth or an eighth of the price of bronze, and its surface can be coloured so as to resemble that metal.

Electro-Metallurgical Treatment of Zinc Ores.—This subject is introduced here, not because the extraction of zinc from its ores by electricity has yet been proved to be a commercial success, but because the subject is of great importance and worthy of special attention. In all processes hitherto employed for the extraction of zinc the loss of metal is great compared with that incurred in the treatment of the ores of other metals, so that an electrical method

would have a greater chance of success with zinc ores than any others.

In order to separate a metal by electrical means, it is essential that it should be in solution. This is usually easily effected. The most abundant ore of zinc is blende (sulphide of zinc), which on being roasted is converted partly into sulphate and partly into oxide, some of the sulphur going off as sulphurous acid. The sulphurous acid may be collected and converted into sulphuric acid in the ordinary way, and used to dissolve the oxide of zinc, or it may be made to act directly upon oxide of zinc or carbonate (calamine), causing the formation of sulphite of zinc, which is also soluble. Having obtained the zinc in solution, it only remains to separate it by means of an electric current and suitable electrodes. For this purpose a positive electrode of lead and a negative of zinc may conveniently be used. Oxide of iron separates at the former and falls to the bottom of the tank, whilst the zinc separated at the latter adheres, and may be removed from time to time, when sufficient has accumulated, by taking out the electrode and putting in another. The appliances required are simple, and the process can be carried on continuously. The success of this method would appear to be assured where a cheap source of power for the generation of the current can be obtained.

ALLOYS OF ZINC AND COPPER.

Brass is usually described as an alloy of zinc with twice its weight of copper, and there is some evidence that the metals enter into a true chemical combination in these proportions ; but, if this be so, the compound is capable of dissolving an excess of either metal, the proportions employed for brass being varied to suit the particular purpose for which it is intended.

Before the year 1780, brass was always made by strongly

heating copper in contact with calamine and charcoal or coal. 3 parts of bean-shot copper (p. 249), 3 of calcined calamine, and 2 of charcoal were heated to bright redness in a covered crucible, when the zinc reduced from the calamine by the action of the carbon, instead of passing off in vapour, as in the extraction of zinc from the ore, entered into combination with the copper, producing brass which was found in a melted state at the bottom of the crucible. Brass is still manufactured by this process at Holywell in North Wales.

When metallic zinc is employed in making brass it is usual to melt the zinc in a crucible, and to add the copper in small portions at a time, until a nearly solid alloy has been produced. This is broken up and remelted with the proper proportion of zinc.

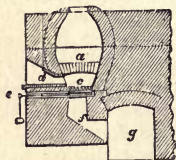


FIG. 86.—Furnace for making Brass. *a*, Arch upon which the crucibles stand. *c*, Grate. *d*, Fire-door. *e*, Damper. *f*, Inclined plane for carrying away cinders into the channel *g*.

The union of the copper and zinc is much facilitated by melting them with old brass. Crucibles made of fire-clay, or of a mixture of fire-clay and plumbago, are employed; convenient dimensions are 16 inches in depth, $9\frac{1}{2}$ inches wide at the mouth, and $6\frac{1}{2}$ inches at the bottom, the sides being 1 inch thick, and the bottom $1\frac{1}{2}$ inch. Eight of these crucibles are heated by a single coal-fire (Fig. 86), the flame of which passes up and circulates around them, and afterwards heats two empty crucibles placed above them, and intended for the subsequent casting of the brass. Suitable proportions of the metals are :

41 lbs. of old brass.
55 lbs. of best selected bean-shot copper.
24 lbs. of zinc.

The crucibles are filled with the pieces of old brass, which are melted down, and leave room for the other metals. Half of the zinc is then introduced, in small lumps, and covered with coal dust; then half of the copper and another

layer of coal-dust ; the rest of the metals is then introduced in the same way, the whole covered with a layer of coal-dust, and the crucibles exposed to the fire for about four hours, when the brass is ready for casting. One of the hot empty crucibles is taken out of the furnace, and placed in another fire so as to keep it red-hot whilst four of the crucibles of brass are emptied into it ; the surface is then skimmed, and the brass is poured into moulds made of slabs of granite mounted in an iron frame, the joints being cemented with clay.

The presence of iron in brass is very objectionable, as it gives rise to deficient tenacity and malleability.

The colour of the alloy of zinc and copper is of course dependent upon the proportions in which the metals are employed. Those alloys which contain more than 80 parts of copper in the hundred exhibit a reddish-yellow colour, in which the red predominates as the quantity of copper increases, the colour becoming yellow when less than 80 per cent. of copper is present. If the amount of copper be less than 30 parts in the hundred the alloy is no longer yellow, but approaches more nearly to the colour of zinc.

The various alloys used to imitate gold, before the art of electro-gilding was introduced, were all modifications of brass. *Dutch metal* or *Dutch leaf gold*, which is one of the most malleable of alloys, is composed of 11 parts of copper and 2 parts of zinc. It is cast into thin plates between slabs of granite, and rolled into sheets, being occasionally annealed (p. 91). When these are very thin, several are passed through the rolling-press together, and they are eventually cut up, and beaten out to extreme tenuity in piles of 40 and 80, under a hammer worked by water-power, making three or four hundred strokes per minute. *Bronze-powder* (or at least one kind of it), used for decorative purposes, is made by reducing the thin leaves of Dutch metal to a fine powder. Such powders are made of different shades, from dark copper colour to pale gold, by varying the proportion of copper. The grinding is effected with a very little oil, to prevent the metal from being tarnished by oxidation.

Pinchbeck is composed of 3 parts of copper to 1 part of zinc, *Prince's metal* of equal weights of the two metals. *Mosaic gold* contains about equal parts of copper and zinc. The same name is sometimes applied to a compound of sulphur with tin.

A little tin is added to the brass intended for engraving, since it causes it to break up more easily under the action of the graver. The addition of a little lead (about 3 oz. to 10 lbs.) much facilitates the working of brass at the lathe and with the file, since it prevents the shavings and filings from *greasing*, or adhering to the tools.

Brass is liable to be rendered very brittle when placed in situations where it is exposed to continual vibration. This seems to be due to the development of a crystalline structure in the metal, and has occasionally caused the snapping of the suspending chains of chandeliers.

The *lacquering* of brass, in order to protect it from being tarnished by the air, consists simply in varnishing it with shellac dissolved in spirit and coloured with saffron, annatto, dragon's blood, &c., so as to give it a golden hue.

Brass is *bronzed* by coating it with a thin film of arsenic, mercury, or platinum, the last being used only for small articles, such as instruments, on account of its high price. A solution of white arsenic (arsenious acid) in muriatic (hydrochloric) acid, or of corrosive sublimate (chloride of mercury) in vinegar, is brushed over the brass, previously warmed, when the zinc in the brass chemically displaces the arsenic or the mercury from the liquid, and one of these metals is deposited as a coating upon the brass. In bronzing with platinum, a solution of muriate of platina (chloride of platinum) is applied in a similar way. There is much art in obtaining a durable bronze coating of any desired shade of colour. A mixture of chloride of platinum, corrosive sublimate, and vinegar, is used for bronzing the *sights* of guns.

Pins which are made of brass wire are tinned by boiling them with granulated tin water, and cream of tartar (bitartrate of potash), when the latter, being strongly acid, slowly

dissolves the tin, which is afterwards displaced from the solution and deposited upon the brass, because the tin and brass, in contact, form a galvanic couple, which decomposes the salt of tin, precipitating that metal upon the surface of the brass, which is the negative plate of the galvanic pair.

In tinning or whitening pins, about 6 lbs. of pins are spread over the bottom of a copper vessel, and covered with 7 or 8 lbs of grain tin ; another layer of pins is then introduced, afterwards more tin, and so on, until the vessel is filled. Water is then poured in, the vessel heated, and $\frac{1}{4}$ lb. of cream of tartar sprinkled over the surface. After boiling for an hour the tinning is completed.

Malleable Brass or Muntz's Metal, or Yellow Sheathing.—This is an alloy of 3 parts of copper and 2 parts of zinc, which differs from common brass in being malleable when hot. It is of course cheaper than ordinary brass, on account of the predominance of the cheaper zinc, and can be more easily rolled into thin sheets. When used for sheathing ships, it keeps cleaner than copper.

The nails employed for securing the sheathing contain, in 100 parts, 87 copper, 4 zinc, and 9 tin, the latter giving them hardness.

Aich Metal, or Gedge's Metal, is an alloy of zinc and copper in nearly the same proportions as are contained in Muntz's metal, but it contains also a little iron. It consists, in a hundred parts, of

Copper	60.0
Zinc	38.2
Iron	1.8

This remarkable alloy is very malleable at a red heat, and may be hammered, rolled, or drawn into wire, with the additional advantage of being readily cast. It has been employed in Austria for casting cannon, and some Chinese cannon have been found to consist of a similar alloy.

*Sterro-metal** is another very strong and elastic alloy used by Austrian engineers for the pumps of hydraulic presses. It

* Named from the Greek adjective, *strong, firm*.

contains copper, zinc, iron and tin, in the following proportions in a hundred parts, the proportions varying between the assigned limits, according to the purpose for which it is required :—

Copper . . .	55 to 60		Iron . . .	2 to 4
Zinc . . .	34 to 44		Tin . . .	1 to 2

Good specimens of *sterro-metal* have been found to offer far more resistance than gun-metal to transverse fracture, and it is only two-thirds of the price. It is said that this alloy was accidentally discovered in an attempt to employ, for the manufacture of brass, the alloy of iron and zinc found at the bottom of the zinc-pots in making galvanised iron (p. 287).

A very hard white alloy of 77 parts of zinc, 17 of tin, and 6 of copper, is sometimes employed for bearings of the driving wheels of locomotives ; and another alloy, containing 90 of copper, 5 of zinc, and 5 of antimony, is used for sockets in which the steel or iron pivots of machinery are to work.

Aluminium-bronze is an alloy of 9 parts of copper and 1 part of aluminium. In colour it much resembles gold, but is much harder and lighter. It is extensively used as a cheap imitation of gold, but it becomes tarnished in course of time. It has also been employed instead of steel for perforating postage-stamps, &c., and is said not to be so soon blunted.

[For alloys of zinc and copper with nickel see *NICKEL*.]

NICKEL AND COBALT.

These two metals almost invariably occur in the same ore. They are so closely allied in nature that their separation from one another is attended with a considerable amount of difficulty. Their properties very nearly resemble those of iron. Both are slightly more fusible than the latter substance ; they are magnetic, but much less so than iron. In respect to tenacity, malleability, and ductility they rival it to the extent that, with our present knowledge, it cannot be

stated which possesses these properties in the greater degree. Nickel and cobalt are both whiter than iron; nickel has a slight yellowish tinge, whilst cobalt has a bluish shade in it. The atomic weight of nickel is given as 58·8, and that of cobalt as 58·7.

The following specific gravities have recently been determined in the author's laboratory :—

	I.	II.	III.	Mean
Grain nickel (a) . . .	7·942	8·044	—	7·993
Malleable cast nickel (b) .	8·364	8·375	8·333	8·357
Malleable nickel, rolled } and stamped (c) . . . }	8·77	8·688	—	8·729

(a) Two specimens of same sample.
 (b) Three
 (c) I. a spoon; II. a fork. "

The specific gravity of cobalt after fusion is stated to be 8·7. Rammelsberg gives its mean specific gravity as 8·957.

Nickel and cobalt behave towards carbon in the same way that iron does. The principal ores of nickel are *Kupfer-nickel* and silicate of nickel. *Kupfer-nickel* is an arsenide of nickel (NiAs), found in the Erzgebirge, near Schneeberg and Freiberg, in Westphalia, Hessia, Bohemia, Thuringia, Hungary, Norway, and Bolivia. It is of a pale copper colour, which becomes black on exposure. *Kupfer-nickel* is usually associated with some or all of the following substances: cobalt, silver, antimony, iron, copper, and sulphur, in variable quantities; nickel also occurs in small quantities combined with sulphur or antimony.

Silicate of nickel has within recent years been found in large quantity in New Caledonia. It occurs in the form of an hydrated silicate of nickel and magnesia, free from arsenic and sulphur, containing about 7 to 10 per cent. of nickel; there is usually some oxide of iron present, mechanically mixed.

The cobalt ores are essentially the same as those of nickel, with the exception of the New Caledonian ore, which is practically free from cobalt.

Nickel and cobalt are extracted from the ores in which they are contained, as sulphides, arsenides, and antimonides,

by concentrating them in the dry way in the form of an arsenide called *speise*, the copper at the same time being obtained in a separate layer in combination with sulphur, forming a *regulus*. These ores being very complex, a considerable amount of experience is required to treat them successfully, for although the tendency is for the nickel and cobalt to unite themselves to arsenic in preference to sulphur, and the copper to combine with sulphur, yet some of the nickel and cobalt will be found in the *regulus* and some of the copper in the *speise*. The presence of iron, antimony, silver, lead, zinc, bismuth, &c. still further complicates the process. Sulphate of baryta (heavy spar) may, when it is required to add sulphur, be conveniently employed for the purpose, as it forms an easily fusible slag, and does not introduce iron into the *speise*.

The following is a general description of the treatment of *speise* at the works of the well-known nickel and cobalt refiners, Messrs. Henry Wiggin & Co., Birmingham. The arsenide is roasted in reverberatory furnaces, in order to drive off in the form of arsenious acid as much as possible of the arsenic, which is condensed in coke-towers, and subsequently used up in the works. The calcined ore is treated with hydrochloric acid in large earthen receptacles, and the solution, containing nickel, cobalt, copper, iron, arsenic, and, perhaps, lead and bismuth, &c., is ladled out into wooden vats. The residue is again calcined and lixiviated and finally resmelted, as it still contains 1 or 2 per cent. of nickel. The solution is diluted and sesqui-oxidised by the addition of chloride of lime (bleaching powder), the quantity added having been adjusted to the iron by analysis. Milk of lime is also added in suitable proportion, to throw down the whole of the sesqui-oxide of iron, which at the same time carries the arsenic with it in combination, as basic arseniate of iron. The precipitate is transferred to a filter, consisting of flannel strained over a wooden frame, and there well washed. It contains sufficient arsenic to make it worth while to use it in smelting operations on

the works. The solution is next treated with sulphuretted hydrogen, which precipitates the copper, bismuth, and lead, as sulphides. Only nickel and cobalt remain to be dealt with. The cobalt is thrown down at a higher temperature as sesqui-oxide by the addition of bleaching power, and the nickel subsequently as hydrated oxide by means of milk of lime.

Oxide of cobalt imparts a fine azure-blue colour to glass, pottery, and enamels, even one-thousandth part producing an appreciable amount of colour.

The oxide of nickel can readily be reduced to the metallic state by making it into a paste with flour or oil, breaking it up into small pieces, and heating it to a bright red in crucibles with charcoal. The product is known as *grain* nickel.

The New Caledonian ore, a pale green silicate, is stated now for the most part to be smelted in blast-furnaces on the spot, much in the same way that iron is treated, and the metal exported to this and other countries in the form of pigs, containing about 5 per cent. of carbon and silicon, and occasionally a little antimony. The difference in cost for carriage between the ore and the pig amounts to about 20 per cent. The New Caledonian pig is refined without expense for arsenic by mixing it with a suitable proportion of arsenical ore.

The New Caledonian ore is sometimes treated in the neighbourhood of Birmingham in the following manner, as are also other ores and products containing nickel, and free from arsenic :—The ore is first fused with iron pyrites, by which means the nickel is separated in the form of a sulphide. The regulus of mixed sulphides is next treated for the separation of iron and sulphur, much in the same way as in the Swansea method of copper smelting, *i.e.* by a series of calcinations and smeltings, the 'metal' in some cases being finally roasted in a fused state, or treated with nitre to remove the last traces of iron. All the iron having been removed, the remaining sulphur is easily got rid of, and the more or less pure oxide of nickel thus obtained is reduced

to the metallic state in the same way as already described in treating of the wet process. It might obviously be rendered purer, if required, by treatment by wet processes such as have already been described. In any case this would be unnecessary when the New Caledonian ore alone was used.

Grain nickel is largely used for making *German silver*—a white alloy of copper, zinc, and nickel, used as a cheap substitute for silver, or for silver-plating on. The relative proportions of the constituents of German silver, or nickel silver, as it is also called, vary according to the purpose to which it is to be put. The commonest made contains about 8 copper, 2 nickel, 3.5 zinc. With a lower amount of zinc the alloy would tarnish very rapidly. With the same quantities of copper and zinc as in the foregoing, and 3 parts of nickel instead of 2 parts, a high-class German silver is obtained, corresponding in colour to silver a little below standard. When the proportion of nickel is increased to 4, the alloy is known as *electrum*; it has a bluish shade in it, and is said to tarnish less readily than silver. The proportion of nickel in this alloy cannot, it is stated, be increased beyond 6 parts without injuring its mechanical properties.

For the purposes named the following proportions are said to be used: (a) Forks and spoons—2 copper, 1 nickel, 1 zinc; (b) knife and fork handles—5 copper, 2 nickel, 2 zinc; (c) for rolling into sheets—3 copper, 1 nickel, 1 zinc. For castings, *e.g.* candlesticks, spurs, bells, &c., 3 parts of lead should be added to the alloy (a), or 2 parts to the alloy (b). The addition of about 1 per cent. of iron to these alloys renders them much whiter, but it makes them harder and more brittle. The iron must first be alloyed with the copper by fusion under charcoal, the product being subsequently alloyed with the remainder of the ingredients. Nickel itself cannot be satisfactorily alloyed with the other constituents of German silver by fusing them altogether. It is best done by melting the copper and nickel together first, and then adding the zinc, either unalloyed or alloyed with copper also. Nickel silver, after cooling from a state of fusion, has

a coarse crystalline fracture ; by heating it to redness and cooling it again the structure is modified, and the alloy can be rolled, and hammered like brass. Coin containing nickel is in use in several countries. German silver was originally made from an ore containing copper, nickel, and zinc, found at Suhl, in Germany.

A white alloy called *arguzoid* has recently been brought out. It is stated to be very tough and to possess mechanical properties superior to brass. Its composition is approximately :—

Copper	56.0 per cent
Zinc	23.0 "
Nickel	13.5 "
Tin	4.0 "
Lead	3.5 "
					<hr/> 100.0

Hitherto nickel has only been used in alloy with other metals as a whitening agent. The reason of this was that it could not be obtained in a workable form. The metal resulting from the fusion of grain nickel is always wanting in malleability and ductility, behaving in a similar way to wrought iron which has undergone fusion, and probably for a similar reason. Dr. Fleitmann in the year 1879 succeeded in rendering nickel malleable by adding to it, whilst in a state of fusion, one-eighth per cent. of magnesium, introduced through a hole in the top of the crucible, a few lumps of charcoal having been previously added. In this way it is possible to produce malleable nickel, which can be welded to iron or steel at a white heat and rolled into thin sheets without separation. Fleitmann failed to get the same results by the use of manganese, aluminium, or calcium. Mr. Alfred Smeaton Johnstone, of the firm of Messrs. Wiggin & Co., who had previously been experimenting with manganese, has, however, succeeded perfectly in rendering both nickel and cobalt malleable by means of manganese. Commercial manganese (about 95 per cent.) or any ferro-manganese may be used for the purpose, the presence of iron not destroying the malleability of the nickel or cobalt. Thus, the analogy

between the treatment of nickel and cobalt and iron which have undergone fusion is perfect. The manganese is added a little at a time to the fused nickel or cobalt, which is kept well stirred during the time, and finally poured out into moulds when tranquil. The metal is considerably agitated by the escape of gas during the addition of the manganese. For most purposes the addition of 2 per cent. of metallic manganese is sufficient; but when the maximum degree of malleability and ductility is required, the quantity added may be increased to as much as 5 per cent., beyond which there is no gain under ordinary circumstances.

Zinc can be successfully alloyed with nickel by reducing their oxides in a state of admixture. By rapidly fusing the alloy thus obtained, a tough, malleable, and ductile metal can be made. The melting-point of nickel is too high to admit of zinc being introduced into it after the nickel is molten. The addition of one-tenth per cent. of magnesium is said to improve the working properties of this alloy.

It is not likely that important applications will be long wanting for a metal possessing such valuable properties as malleable nickel, the price (about 4s. a lb.) being comparatively low. Malleable nickel anodes are already being substituted for the old cast anodes. The former certainly have a great advantage in their uniformity of structure, which not only prevents their being eaten away irregularly, but also avoids loss of energy through secondary currents. Whether the presence of manganese will prove an objection remains to be seen. In any case, with proper care in the manufacture, it could in all probability be reduced to so small an amount as to be innocuous. We are guided to this conclusion as to the possibility of controlling the manganese by our knowledge of the parallel case of Siemens steel.

The greater rarity and consequent higher price of cobalt precludes its general use in the solid metallic state. It is, however, being used for electro-plating, the articles coated with it being sold as superior nickel-plate.

The use of oxide of cobalt as a colouring agent has already been referred to. It is largely employed in the form of *smalt*, a silicate of cobalt and potash. Great care has to be exercised in the manufacture, any impurities deteriorating the quality of the colour. When iron and copper are present, in order to separate them advantage is taken of their greater affinity for oxygen; whilst nickel can be separated by reason of its having a greater affinity than cobalt for arsenic, so that the cobalt can be obtained as silicate and the nickel as speise in two distinct layers.

LEAD.

Whether lead in the metallic state has ever been found as a true natural product appears to be doubtful, since the small quantities which have been found associated with the ores of lead may have been accidentally reduced.

Although minerals containing lead are pretty abundant, there are only two which are found in sufficient quantity to serve as sources from which to extract the metal on the large scale.

Ores of Lead.

	Composition	Lead in 100 parts of the pure ore
Galena or Sulphuret of Lead } .	Lead, Sulphur	86 $\frac{1}{2}$
White Lead Ore or Carbonate of Lead } .	Lead, Oxygen, Carbonic Acid	77 $\frac{1}{2}$

Galena is by far the most abundant of the compounds of lead. It forms extensive veins, traversing clay slate in Cornwall, and limestone in Derbyshire and Cumberland. It is also found in Flintshire (Holywell), Scotland (Leadhills), and the Isle of Man. Spain yields abundance of galena in Catalonia, Grenada, and at Linares in the Sierra Morena, where it occurs in granite. This ore is also abundant in the Upper Hartz, at Freiberg in Saxony, and in the United

States of America. Few ores are so easily recognised at once as galena; it is distinguished by its lustre, which is almost metallic, its dark grey colour, and its great weight (specific gravity, 7.5). It can generally be easily split up into rectangular fragments, and often occurs in distinct cubical crystals of large size.

Galena almost invariably contains silver, which takes the place of a part of the lead in its combination with sulphur, without producing any alteration in the crystalline form and general appearance of the ore. A galena containing two parts of silver in a thousand would be spoken of as an *argentiferous galena*,* because even that small proportion of metal can be profitably extracted from the lead after smelting it from the ore.

Antimony is also found in many specimens of galena, as a sulphuret of antimony, and its presence has a serious influence upon the quality of the lead extracted from the ore. The minerals commonly associated with galena in the vein are blende (sulphuret of zinc) and copper pyrites, whilst *cawk* or heavy spar (sulphate of barytes), calc-spar (carbonate of lime), and fluor-spar (fluoride of calcium) are often found adjacent.

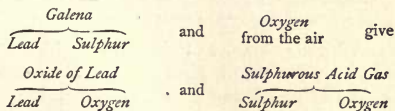
White-lead ore or *carbonate of lead* is a much less important ore, often occurring in veins of galena, and apparently produced by a chemical alteration of this ore. When pure, it is a white crystalline mineral, but it has often an earthy appearance, and it is so unlike galena that miners have been known to reject it as worthless. Sometimes it has a dark colour, from the presence of a little galena intermixed with it. Carbonate of lead is found in considerable quantity near Aix-la-Chapelle, as well as in Spain, and in the valley of the Mississippi. This ore is so seldom smelted apart from galena, that it is not necessary to describe its treatment separately.

* *Argentum*, Latin for silver; *fero*, I bear.

Sulphate of lead (composed of lead, sulphur, and oxygen), or *Anglesite*, is very rarely found in any quantity. Australia furnishes some of it, containing a considerable proportion of silver.

In order to prepare the lead-ore for smelting, it is sorted by hand, the worthless pieces being rejected, and broken up, either with a hammer or between crushing-cylinders; it is then washed, in much the same way as the ore of tin (p. 251), in order to separate, as far as possible, the foreign matters mingled with it. The differences in the ore have led to the adoption of different methods of conducting the operation of smelting; thus in Derbyshire and Flintshire, where the lead ores are rich and contain very little quartz (silica), the galena is smelted in reverberatory furnaces, whilst at Alston Moor, and generally in the lead-works of the North, small blast furnaces are employed.

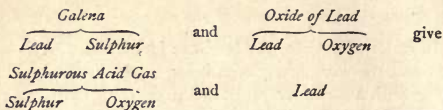
Smelting of Galena in the Reverberatory Furnace.—The chemical principles upon which metallic lead is separated from galena are similar to those involved in the last stage of the extraction of copper (roasting the fine metal for blistered copper, p. 238), the sulphur being finally expelled in the form of sulphurous acid, produced by its combination with oxygen previously taken up from the air. The galena is first roasted until a part of it has become converted into oxide of lead, its sulphur having combined with oxygen and been removed as sulphurous acid.



During this roasting process, another portion of the galena is converted by the oxygen of the air into sulphate of lead.

When the roasting has proceeded far enough, the oxide of lead and sulphate of lead are melted with that portion of the

galena which has escaped alteration, when the whole of the sulphur is converted into sulphurous acid, and the lead is left in the metallic state.



Again—

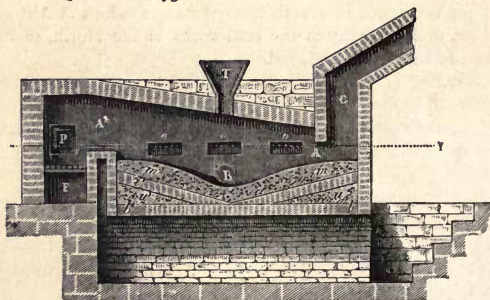
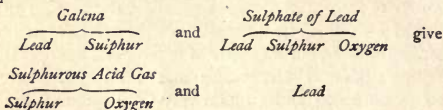


FIG. 87.—Reverberatory Furnace for smelting Galena.

The reverberatory furnace in which the smelting of galena is effected is represented in Figs. 87, 88. The hearth (B) is about eight feet by six, and is separated from the grate (F) by a fire-bridge which rises to within about eighteen inches from the arch (AA'), the latter gradually descending, as it approaches the chimney, until it is within about six inches of the hearth. The flame and products of combustion, after passing over the hearth, are conducted by two openings into

a flue about eighteen inches wide ; this flue makes a bend downwards towards the top, and is carried into a chimney between fifty and sixty feet high. The flue is so constructed that it may be readily opened to clear out the deposit from the lead fumes. The fire-door (P), for throwing the coal upon the grate, and the ash-pit (F) are on opposite sides of the furnace ; that upon which the fire-door is situate is called the *labourer's side*, whilst that opposite is the *working side*.

On the labourer's side, there are three openings (o), about six inches square, at equal distances, which can be closed when necessary with iron plates. There are three corre-

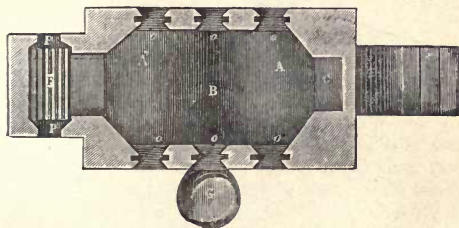


FIG. 88.—Plan of Reverberatory Furnace for smelting Galena.

sponding openings on the working side of the furnace, as well as two tapping-holes for the lead and slag respectively.

The hearth of the furnace is lined with the slags from previous operations, which are spread over it while in a pasty state, before solidifying, and fashioned to the proper shape as shown in Fig. 89. On the labourer's side, it is nearly up to the level of the working doors, but on the opposite side it is hollowed out so as to be eighteen inches below the middle door ; this being the lowest part of the hearth, where the melted lead collects, a tap-hole is provided for running off the metal, and at some distance above it is the aperture for the escape of the slag. Adjacent to the tap-

hole there is a basin outside the furnace for the reception of the lead. The Derbyshire ores commonly contain heavy spar (sulphate of baryta), which, owing to its high specific gravity, cannot be separated by washing from the galena. If the ore does not already contain it, fluor-spar is added to flux the heavy spar.

The operation of smelting galena in the reverberatory furnace consists of four consecutive stages, distinguished as first, second, third, and fourth *fires*.

First fire.—As soon as the lead smelted in the preceding

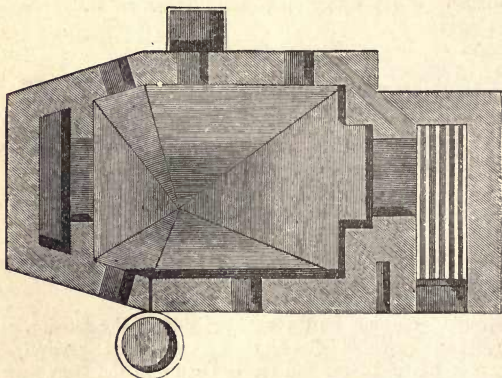


FIG. 89.—Hearth of Reverberatory Furnace for smelting Galena.

operation has been tapped into the outer basin, and while the furnace is still glowing, the fresh charge of about a ton of ore is introduced through a hopper (τ , Fig. 87) in the arch of the furnace. No regular fire is made up, but only a little coal is thrown into the grate to keep up a moderate temperature, for if this were raised too high at first, the galena would fuse, and the roasting would be rendered impossible. A workman stationed at the labourer's side spreads the ore uniformly over the surface with a rake, after

which the doors are closed and the draught moderated by lowering the damper. Since there is little fuel upon the grate, a considerable quantity of unconsumed oxygen of the air passes over the hearth of the furnace, so that some oxide of lead and sulphate of lead are soon formed.

After the roasting has been continued for some time, the skimmings from the lead, run into the outer basin at the end of the last smelting, are thrown into the hearth. These skimmings consist chiefly of a mixture of sulphur with a large proportion of lead, which is thus rendered less fusible. This dross is speedily acted on by the oxide and sulphate of lead, as above explained, with separation of metallic lead which runs down into the hollow, and is drawn out through the tap-hole into the basin. This first portion of lead contains a larger proportion of silver than that tapped at a later period of the process. The workman occasionally turns over the ore to expose fresh surfaces, and, if necessary, throws a little small coal upon the charge to prevent the oxidation from being carried too far. About an hour after the commencement, a large quantity of lead is run off, being chiefly derived from the action of the skimmings upon the roasted galena.

After an hour and a half from the commencement, all the doors are thrown open, and the ore is well turned over by two workmen placed on opposite sides of the furnace, after which the doors are closed. At the end of two hours, the first fire is completed, a sufficient proportion of the galena having been converted by the roasting into oxide and sulphate of lead.

Second fire.—The damper of the furnace is now partly raised, and more coal is thrown into the grate, so as to bring the temperature up to a bright red heat.

The sulphuret of lead in the ore now acts upon the oxide and sulphate formed during the previous roasting, and the melted lead begins to run out in abundance. The workman stationed on the working side thrusts the pasty slags out of

the basin, whilst the man on the labourer's side spreads them over the rest of the hearth; a little quicklime is now thrown in to assist stiffening the unreduced portion. Were the stiffening effected at this stage entirely by means of lime, the unreduced portion would be rendered too infusible for subsequent operations, so it is made sufficiently pasty partly by opening the fire-door and lowering the temperature. The charge is well worked and calcined for about an hour. The thickening of the charge not only allows any reduced lead to drain out, but also—and this is the main object—prevents the galena from sinking below the slag, and being thus removed from the action of the air.

Third fire.—The doors are all shut, and the damper entirely opened, more fuel being thrown upon the grate so as to raise the hearth to a still higher temperature for about three quarters of an hour, when the doors are again opened, the slags spread over the hearth, and a fresh quantity of lime thrown upon them. The lime enters into combination with any silica which may have united with the oxide of lead, and sets the latter free to act upon any portions of unaltered sulphuret of lead. The lime also acts advantageously by diminishing the fusibility of the mass and thus facilitating the contact between the sulphuret of lead and the oxide. This third fire also occupies about an hour.

Fourth fire.—The grate is again charged with fuel and the doors closed for about three quarters of an hour, the furnace being thus raised to its highest temperature. The tap-hole is then opened to allow the lead to run into the outer basin (G, Fig. 88), and some lime is mixed with the slags in order to *dry up* or partly solidify them, when they are raked out through the openings on the labourer's side, and the furnace is ready to receive a fresh charge of ore. A little small coal is sometimes thrown upon the hearth at the conclusion of the fourth fire, to remove the oxygen from any oxide of lead which may still remain.

The iron of the tools employed in stirring the contents of the hearth is seriously corroded by the sulphur in the ore.

The whole operation of smelting in the reverberatory furnace lasts about five hours, and the coal consumed is about 12 cwts. for every ton of ore.

The rich scum which forms on the metal in the pot contains a considerable quantity of lead mechanically intermixed; to effect the separation of which the workman agitates the lead vigorously with a paddle, at the same time throwing some coal slack into the pot and igniting, by means of a shovelful of hot cinders, the gases thus generated. In this way the dross is prevented from solidifying and the greater part of the entrapped lead separated.

The slag amounts to about one-fourth of the weight of the ore, and sometimes contains as much as 40 parts of lead in the hundred, so that it is smelted in the *slag-hearth*, to be described hereafter.

At Bleiberg in Carinthia, the lead is extracted from galena by a process much resembling that just described; but wood is employed as fuel, the grate being at the side instead of at the end of the hearth, and the hearth of the furnace is a single inclined plane, allowing the reduced lead to flow at once out of the furnace. The first portion of lead which runs out is known as *virgin lead*, and is purer than the *pressed lead* obtained later in the process when the temperature is much higher. The Carinthian lead (*Villacher lead*, from the town of Villach) is in high repute for its purity.

In Nassau, a furnace more nearly resembling the English reverberatory is employed, and towards the end of the process, some green wood is added to the charge upon the hearth, in order that the steam and gases evolved from it may agitate and mix the pasty mass. The lead collected in the basin outside the furnace is stirred with wood (like tin, see p. 259) before being run into pigs.

In some of the Continental furnaces, metallic iron is added to the charge in order to combine with the sulphur in the galena, and separate the lead in the metallic state.

When galena containing much antimony is smelted in the reverberatory furnace, a portion of the oxide of lead com-

bines with the oxide of antimony to form a compound which can only be decomposed by the coal at a very high temperature, so that the first portions of lead obtained are much purer from antimony than those at the end of the process.

Smelting of Lead Ore in the Scotch Furnace or Ore-hearth.—Since this is a blast furnace, it is found advantageous to roast the ore before smelting it, in order that it may be rendered more porous and may offer less obstruction to the blast.

The ore is spread over the hearth of a reverberatory furnace, not unlike that employed for roasting copper ores (p. 229), in charges of about half a ton, and roasted at a moderate heat for about eight hours, being frequently turned over as is usual in roasting operations. Some antimony is thus expelled from the ore, which would otherwise harden the lead; a considerable quantity of sulphur also burns off. The roasted ore is raked out of the furnace into a pit filled with water, which causes it to fly into fragments suitable for charging into the ore-hearth.

The ore-hearth (Fig. 90) is a small square forge or blast furnace about two feet high, and 18 inches by 12 internal area. It is arched over at the top, so that the *lead fume* may be conducted into a long flue, sometimes five feet high and three feet wide, in which a large quantity of oxide of lead and sulphate of lead is deposited. At the Allenheads works, this flue is carried up the side of a hill for three miles before it terminates in the chimney, in order to secure perfect deposition of the lead fume, which would otherwise involve very considerable waste; for although lead is not, like zinc, a metal capable of being distilled, both the metal and its sulphuret, when heated in a strong current of air, are liable to be carried off in the form of vapours, which afterwards combine with oxygen from the air, and are deposited as oxide and sulphate in the flues; these deposits are afterwards heated in the calcining furnace till they can be

made to stick together, and are then smelted in another furnace called the *slag-hearth*. A *rain chamber* is often provided, in which the condensation of the fume is assisted by water ; and the great length of flue renders it necessary to assist the draught by large exhausting pumps.

Since a very moderate temperature is required in this furnace, the sides and bottom are lined with cast-iron plates, and in front of the furnace, where there is an opening about

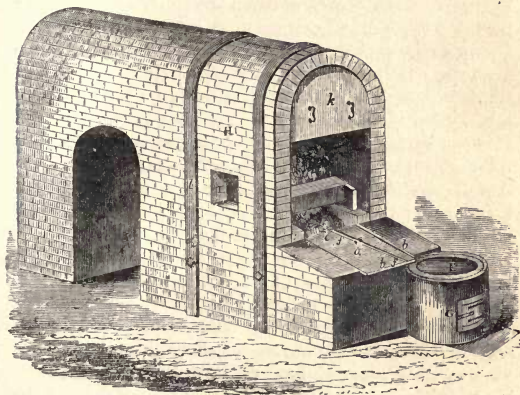


FIG. 90.—Scotch Furnace or Ore-hearth for the extraction of Lead.

a foot high, a sloping iron plate *ab* (*work-stone*) is fixed, upon which the materials can be raked out when necessary, for examination and manipulation by the smelter. The bottom of the furnace, upon which the melted lead collects, is about $4\frac{1}{2}$ inches below the upper surface of this iron plate, in the edge of which there is a groove cut, near to the side of the furnace, through which the melted lead may run when it rises to a sufficient height, into a gutter (*f*) which conveys it

into a cast-iron pot (f) heated by a separate fire, and called the *melting-pot*.

The blast-pipe enters at the back of the furnace, about 11 inches from the bottom. Peat is the principal fuel employed in the furnace, in the form of square blocks, with which the furnace is filled at the commencement, some judgment being required in their arrangement, and the fire is lighted by placing one of the blocks, already kindled, in front of the blast-pipe, when the combustion soon spreads throughout the furnace. The first charge introduced into the ore-hearth does not consist of the roasted ore, but of the residue from a previous smelting operation, which is called *browse*, and consists of partly reduced ore mixed up with cinders; before this is thrown in, a little coal is put on the fire to raise the temperature, and in a short time the charge is raked out upon the work-stone in front of the furnace, and examined, in order that the *grey slag*, a shining glassy mass, may be picked out and thrown on one side. This grey slag contains a quantity of silicate of lead, with silicate of lime, &c., and requires a higher temperature for the extraction of its lead than is attainable in this furnace; it is therefore smelted in the *slag-hearth*, to be presently noticed.

The browse cleaned from slag is thrown back into the furnace, and its behaviour observed; should it appear to melt too readily, it is rendered less fusible by adding a little lime, lest it should run down to the bottom of the furnace with the metallic lead; on the other hand, if it does not become soft enough to permit the lead to separate, lime must also be added to soften it. These apparently opposite effects of the addition of lime are due to the bases not readily fusible *per se* at that temperature being present in excess, thus thickening the slag; whilst, in the second case, they are only in sufficient quantity to form fusible compounds with the silica, &c.

A peat is now placed before the opening of the blast-pipe, in order to prevent any dust from entering it, and a quantity of roasted ore with a little coal is thrown in. After

about twenty minutes, the charge is again raked out on to the work-stone, the grey slag picked out, the remainder thrown back into the furnace, and a fresh charge of roasted ore and coal added.

These operations are repeated during 14 or 15 hours, in which period one or two tons of lead will have collected in the outer basin, according to the richness of the ore, and the proportion (varying from $\frac{1}{10}$ th to $\frac{1}{15}$ th of the whole) which has been removed in the grey slag.

The separation of the metallic lead is partly due to the action of the sulphuret upon the sulphate and oxide of lead, as explained at p. 304, and partly to the removal of oxygen from the oxide by the carbon of the fuel.

The lead extracted in the ore-hearth is purer and softer than that obtained by the reverberatory furnace, the temperature being so low that the other metals contained in the ore are not reduced.

Smelting of Slags, &c., in the Slag-hearth.—In this operation the object is to extract as much of the lead as possible from the slags and other residues, without reference to its purity, by the employment of a very high temperature, so as to completely liquefy the slag. The general construction of the furnace (Figs. 91, 92) is not very different from that of the ore-hearth, but it is larger, being 3 feet high, and 26 inches by 22, internal area. The sides are built of sandstone, in order to resist the much higher temperature of this furnace. The bottom (A) of the furnace consists of a cast-iron plate, and is covered with a layer, about 16 inches thick, of porous cinders tightly rammed down, which serves as a strainer to separate the lead from the slag, since the melted metal easily percolates into the porous cinders, which protect it from being oxydised again by the air, and runs thence into a receptacle (B) outside the furnace, which is also filled up with similar cinders, and has an opening through which the lead flows into an iron pot (E) kept hot over a separate fire. The slag runs off the surface of the

layer of cinders, both in the furnace and in the receiving basin, and falls into a cistern of water (c), where it breaks up, so that the lead entangled in it is easily separated by washing. The fire is lighted with peats as in the ore-hearth, the

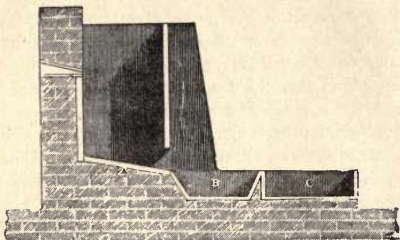


FIG. 91.—Slag-hearth for extracting Lead.

blast being forced through a nozzle at the back of the furnace, about four inches above the layer of cinders. Some coke is then thrown in, and about six hours after, when the temperature is sufficiently high, a charge of the slags, &c., which are to be smelted. Coke and slags are thus added in

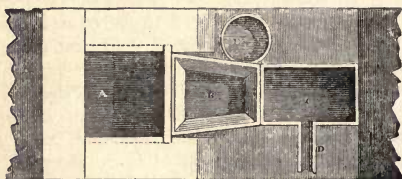


FIG. 92.—Plan of Slag-hearth.

alternate charges, as in the iron blast-furnace, until the furnace requires repair.

The charge for the slag-hearth commonly consists of :

- 100 parts of slag from the reverberatory furnace ;
- 20 „ coal-ashes ;
- 13 „ clay-hearths of old furnaces, impregnated with lead ;
- 5 „ rich slag from a previous operation.

The silica and alumina present in the clay and in the coal-ashes combine with the lime and oxide of iron in the slag from the reverberatory furnaces, and form an easily-fusible slag. The lead is reduced to the metallic state mainly by the action of the heated carbon, which removes the oxygen from the oxide of lead. The coke is piled up towards the front, and the charge towards the back of the furnace, and a *nose* or prolongation of the tuyère is allowed to be formed by the solidified slag, so as to carry the blast up the centre of the furnace. When a cold-blast is employed, this nose is apt to become too long, so that air heated to about 300° F. is found to answer better, beside effecting a considerable saving of fuel. If the blast is too hot, the slag will not be chilled so as to form a nose.

A very inferior description of lead (*slag-lead*) is obtained from the slag-hearth.

In some parts of Spain, lead is extracted from the slags of the Roman lead-furnaces.

Richardson's Furnace, or the *Economico Furnace*, which is employed in Newcastle and the neighbourhood instead of the slag-hearth, as well as for the extraction of lead from the ore, is a modification of the *Castilian furnace* (Fig. 93), being also a blast-furnace, in which the blast is either supplied by a blowing-engine through three tuyères or blast-pipes, or is drawn into the furnace through five or six openings, by the action of a small chimney. The body (A) of the furnace is circular and is built of fire-brick, about $8\frac{1}{2}$ feet high and $2\frac{1}{2}$ feet in diameter, the bottom (B) being lined with a mixture of clay and powdered coke, well beaten down, and hollowed out to receive the lead. The ore, or mixture of ore and slag, smelted in this furnace, must not contain more than 30 parts of lead in the hundred. The ore is roasted previously to its introduction into the furnace, which is charged with ore and fuel through an opening (1) in the square brick structure supported on four pillars, which surmounts the furnace, and, if necessary, the charge is sprinkled with water from a rose, to

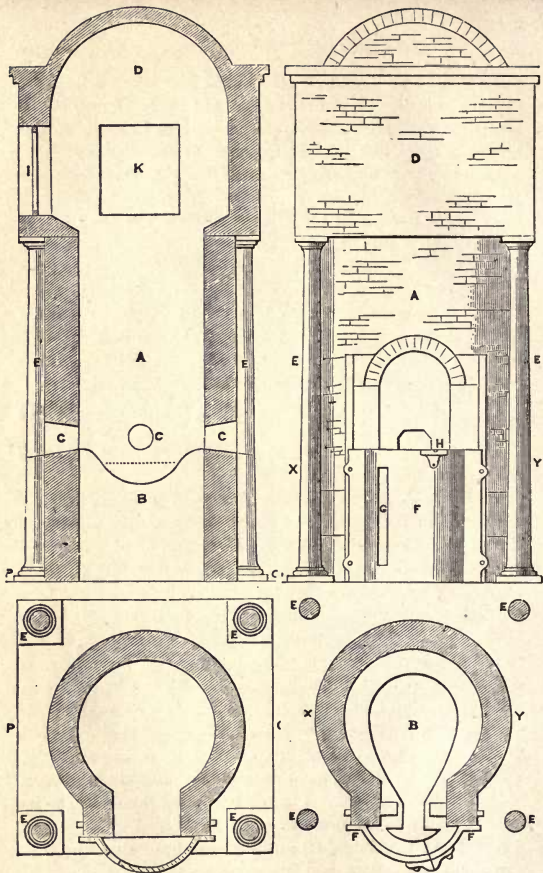


FIG. 93.—Castilian Furnace for Lead-smelting. A, Body of the furnace. B, Crucible for receiving the lead. C, Blast-pipe. D, Masonry enclosing the top of the furnace. E, Cast-iron pillars. F, Receptacle for the lead. G, Slot for the tapping-hole. H, Spout for overflow of slag. X, Charging-door. K, Flue. P, Ground line.

prevent the dust from being carried into the flue. The slag flows over the side of the hearth, as in an iron blast-furnace, into cast-iron waggons, whilst the lead accumulates in the cavity at the bottom, and is tapped out from time to time into an iron basin (F). Limestone is sometimes mixed with the charge, to flux the siliceous matters.

At Clausthal in the Hartz, the galena is reduced by fusing it in a small blast-furnace or *cupola-furnace* with granulated cast-iron, which combines with the sulphur to form a sulphuret of iron, and sets the lead at liberty. The sulphuret of copper which is present in the ore is not decomposed in the process, but forms a *matt* upon the surface of the lead, and after converting the sulphuret of iron into oxide by roasting, and removing the oxide by fusion with siliceous matters, the sulphuret of copper is sent to the copper smelting-works. At some works, slags from the refining of iron (p. 163) are employed to assist in the decomposition of the sulphuret of lead. When ores in a finely divided state form part of the charge of the slag-hearths and cupola furnaces on the Continent, they are often mixed with clay or lime and moulded into bricks before being thrown into the furnace.

Softening of Lead in the Calcining or Improving Furnace.—The lead obtained by either of the above processes sometimes contains considerable quantities of silver, antimony, copper, and iron, which harden the metal and render it unsuitable for some of its applications. English lead is the purest which is to be found in commerce, and Spanish lead is the most impure; the composition of two samples in 100 parts is here contrasted:

	English	Spanish
Lead	99.27	95.81
Antimony	0.57	3.66
Copper	0.12	0.32
Iron	0.04	0.21

Even the English specimen is sufficiently impure to be designated a *hard lead*.

When the lead contains any notable proportion of silver, it is treated by a special process to be described hereafter, but when the hardness is due to the presence of antimony, &c., the lead is softened or improved by exposing the melted metal in a very shallow pan to the action of the oxygen of the air, which converts the antimony, copper, iron, and a considerable portion of the lead, into oxides, which collect as a dross upon the surface, and are skimmed off at intervals, until the lead is found to be sufficiently softened.

The improving furnace (Figs. 94, 95) is a reverberatory furnace with a low arch, 18 inches above the hearth near the fire-bridge, and 6 inches near the chimney, towards which the flame is drawn by two flues (F). Since a large

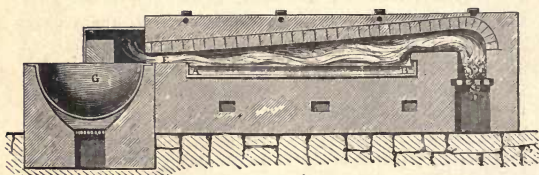


FIG. 94.—Calcining Furnace for improving Hard Lead.

spreading flame is required, the fireplace (D) is 5 feet long and 18 inches wide, being divided from the hearth by a fire-bridge 27 inches wide and 16 inches above the hearth (B). In the hearth of the furnace there is set, with a space round it to allow for expansion, a cast-iron pan measuring 10 feet by 5, which is 8 inches deep at the end nearest the grate, and 9 inches at the other end; at this deepest end there is an iron gutter (f) stopped up during the process, by an iron plug with a weighted lever through which the lead may be run out when it is sufficiently refined.

Eight or ten tons of the hard lead are melted in an iron pot (G) and ladled into a gutter (H) through which they run into the improving pan, which has been already heated to

dull redness; the gutter is then closed by a damper, and the improving process commences. Its duration depends upon the amount of the foreign metals (especially of antimony) which the lead contains, a single day's calcination being sufficient to soften some leads, whilst others require two or three weeks' exposure to the action of the air which passes through the furnace. The dross is raked off occasionally, so as to expose the surface of the metal, and the progress of the refining is observed by ladling a small sample into an ingot-mould, when its surface assumes a peculiar crystalline appearance if the refining is completed. The

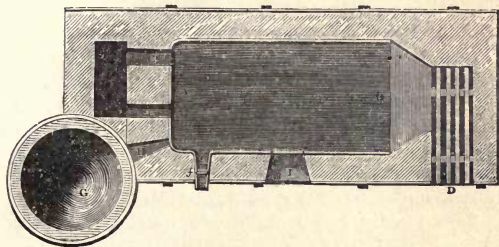


FIG. 95.—Plan of Improving Furnace.

refiner also judges that the lead is sufficiently softened, if a rainbow or iridescent film is formed upon the surface when a rake is pushed over it from the working door. The softened-lead should not form round globules when poured upon a heated iron plate.

The dross, which consists chiefly of the oxides of antimony and lead, is mixed with coal and ground under edge-runners, previously to being smelted in a small reverberatory furnace, the hearth of which gradually slopes down towards the chimney, where there is a cavity for the reception of the metal, which constantly flows out through the tap-hole into an iron pot, to be afterwards transferred to the pig-moulds.

The hard lead thus obtained is either again calcined with a fresh portion of metal, or if it contains a very large proportion of antimony (of which some specimens contain a third of their weight), it is sold to the type-founders.

The process of improving the moderately hard lead produced at Altenau in the Upper Hartz resembles the *boiling* of tin, and consists in melting about 11 tons of the metal in an iron pot 5½ feet deep and 3 feet wide, and stirring it for two hours with a birchen pole moved by machinery, when the violent bubbling of the gases through the metal continually renews the surface in contact with the air, causing the formation of dross containing the impurities.

In France, the improving furnaces often have two fires, one at each end of the pan.

Another method of improving very hard lead consists in melting it, as above, in a cast-iron pan, and throwing upon the skimmed surface a small quantity of a mixture of Peruvian saltpetre (nitrate of soda), soda and lime, the addition being repeated until the metal is sufficiently softened; the oxygen of the saltpetre converts the antimony into antimonious acid which combines with the soda and lime, and is removed as dross, together with a considerable quantity of oxide of lead.

Refining of Lead containing Silver by Pattinson's Process.—

This very simple and beautiful process, which was introduced in 1829, has not only greatly improved the quality of lead, but has very much increased the production of silver in England; previously to that date no process existed by which the silver could be removed so as to leave the lead in the metallic state, and it was necessary to convert the whole of the lead into an oxide in order to separate the silver, this oxide being afterwards smelted to recover the lead. Since this could not be made to pay unless the lead contained at least eleven ounces of silver in the ton, any smaller quantity of silver was left in the lead sent into the market, the lead being thereby hardened, and the silver entirely lost for all

useful purposes. After the introduction of Pattinson's process, much of the old lead was eagerly bought up for the sake of the silver which it contained.

The process depends upon the property of lead to crystallise at a lower temperature than an alloy of lead and silver, so that if melted lead containing a small proportion of silver be allowed to cool slowly and constantly stirred, the small crystals of lead which are formed at first will contain little or no silver, that metal remaining in the liquid portion.

To carry out this principle, a series of melting-pots is employed. The number of pots as well as their form and general arrangement will differ somewhat in different esta-

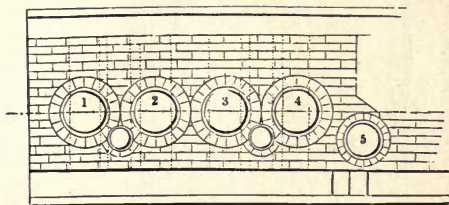


FIG. 96.—Pots for desilverising Lead.

blishments, but for the sake of illustration, a *desilverising* plant containing five pots may be taken (Fig. 96). These pots are made of cast iron and set in masonry; the *working-pots*, 1, 2, 3, and 4, are oval in shape, their mouths being 40 inches by 26, and they are shaped at the bottom like the small end of an egg. 5 is the *market-pot* for melting the desilverised lead before casting it into pigs; it is smaller than the others. The smallest pots (about two feet in diameter), between 1 and 2, and between 3 and 4, are the *temper-pots*, for containing the melted lead in which the perforated ladle (Fig. 97) is warmed, which is used for fishing out the crystals of lead. This is an iron ladle about 18 inches wide and 5 inches deep, with an iron handle of 4½

feet and a wooden handle of about 5 feet in length; the holes in the ladle are $\frac{1}{2}$ inch wide and $\frac{3}{4}$ inch apart. Each pot is heated by a separate fire.

The lead to be refined is usually in pigs (or *salmons*) weighing from 120 to 140 lbs. each. About 64 of these (or 4 tons) are melted in pot 1. (See Fig. 98.) When they are perfectly melted, the fire is raked out, and the oxide is skimmed from the surface of the lead. In order to hasten the cooling of the metal, one or two pigs of cold lead are thrown in, or a little water is thrown upon the surface so as to form a solid crust, which is then pushed down into the liquid metal. This is continued until crystals of lead begin to form.

FIG. 97.—Ladle for fishing out Lead-crystals



The workman then detaches any lead which has solidified on the sides of the pot, and stirs the melted metal with an iron bar in order to preserve an equal temperature throughout. Another workman takes the perforated ladle out of the temper-pot in which it has been heated, and fishes up the crystals which have formed. The handle of the ladle is then rested upon a pig of lead faced with iron placed at the edge of the pot to serve as a fulcrum, and the workman seizes the end of the long handle, and jumps down from the platform around the pots on to the floor, thus tilting the ladle up out of the melted lead, over which he shakes it violently so as to drain all the liquid metal back into the pot. The ladle is then swung by a crane over pot 2, into which the crystals are thrown; after this has been repeated for an hour, only about one ton of lead richer in silver is left in pot 1, the quantity being ascertained by trying the depth of the metal in the pot.

The removal of the crystals is still proceeded with, but

since these will now contain too much silver to be introduced into pot 2, they are thrown upon the ground in order to be afterwards melted up with more lead in pot 1. When only $\frac{1}{2}$ ton of the rich liquid alloy is left in pot 1, it contains about three times as much silver as the original lead, and is ladled out and cast into eight pigs, which often contain as much as 150 ozs. of silver in the ton, together with any copper and

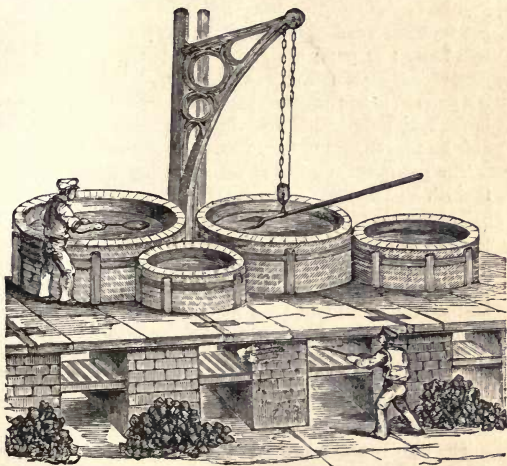


FIG. 98.—Desilverising Lead by Pattinson's Process.

antimony which were contained in the original lead; whilst any arsenic which was present will have passed into the crystals. The half-ton of crystals which have been thrown upon the ground are now melted in pot 1 with a fresh quantity of the original lead, and treated as before.

The three tons of crystals of lead poor in silver which were transferred to pot 2, are made up to four tons by adding

lead of the same richness in silver, and submitted to a repetition of the same treatment, about three-fourths of it being transferred, in crystals, to pot 3, one-eighth of it, in the form of richer crystals, being thrown upon the ground to be remelted in pot 2, and the remainder, which is left in the liquid state at the bottom, is ladled out into pot 1.

The crystals in pot 3 are treated in the same way, the portion remaining liquid being transferred to pot 2, and the poorer crystals melted in pot 4. Finally the crystals of poor lead formed in pot 4 are ladled into pot 5, to be cast into pigs, which are treated again, if necessary, until the silver is at last reduced to half an ounce in the ton. By a single operation in the four pots, as just described, the silver in the marketable lead is reduced to one-tenth of its original amount.

The process of cupellation for extracting the silver from the rich lead is described under Silver.

The quality of the lead is greatly improved by Pattinson's process, not only because the bulk of the antimony and copper remain with the silver in the liquid portion, but because these and other impurities which tend to harden the lead are converted into dross by the oxygen of the air, precisely as in the improving or calcining process.

The description just given refers to an operation with Pattinson's process, upon the *low system*, as it is termed, in which $\frac{7}{8}$ ths of the contents of each pan are removed in crystals, whilst according to the *high system* only $\frac{2}{3}$ ds are removed.

The high system is preferred for the treatment of the leads richer in silver and of otherwise impure quality, as many as fifteen pans being sometimes employed, so that there is a greater chance of oxidising the impurities. Both systems are combined in some establishments, in order to suit the different descriptions of lead. In working the high system, it is generally found that the crystals taken out of a given pot contain about half as much silver as the alloy originally contained; thus nine tons of a silver-lead containing ten ounces of silver to the ton would yield six tons of crystals

containing only five ounces to the ton, and three tons of liquid alloy containing twenty ounces to the ton.

Where many pans are employed, they are commonly hemispherical, being about 5 feet wide and $2\frac{1}{2}$ feet deep.

If the lead contains as much as 60 ozs. of silver in the ton, the third pan is selected for melting it, but if it contains only seven ounces in the ton, it is melted in the 7th pan, an intermediate pan being selected in other cases.

As much as twelve tons of lead are sometimes melted, to begin with, upon the high system, and two-thirds of it are ladled out in crystals into the next left-hand pan—an operation requiring about two hours. The liquid alloy is then chilled into a pasty condition by throwing water upon it, and ladled out into the next right-hand pan, the melting-pot being again charged with twelve tons of the original lead; after a sufficient quantity of metal has accumulated in the adjacent pots, the crystallising operation is repeated with their contents, the two-thirds of crystals being always transferred to the next left-hand pan, and the one-third of liquid alloy to the next right-hand pan. Supposing, therefore, the rule to hold good, that the crystals contain half as much, and the liquid twice as much silver as the silver-lead from which they were obtained, the lead would be found, after passing through four pots to the right of the melting-pot, to contain sixteen times as much silver as the original lead, whilst, after it had passed through four pots to the left of the melting-pot, it would contain only one-sixteenth of the original proportion of silver, as indicated below, where a lead containing seven ounces of silver in the ton is supposed to have been melted originally in pot No. 7.

No. 1.	No. 2.	No. 3.	No. 4.	No. 5.	No. 6.
448 oz.	224 oz.	112 oz.	56 oz.	28 oz.	14 oz.
per ton.	per ton.	per ton.	per ton.	per ton.	per ton.

No. 7.	No. 8.	No. 9.	No. 10.	No. 11.
7 oz.	$3\frac{1}{2}$ oz.	$1\frac{3}{4}$ oz.	$\frac{7}{8}$ oz.	$\frac{7}{16}$ oz.
per ton.	per ton.	per ton.	per ton.	per ton.

Instead of waiting until the proper quantity of metal for crystallising has accumulated in a given pot, the right complement of a sample of lead of the richness suitable to that pot is commonly added from the stock kept in the establishment.

A modification of the Pattinson process, in which hand-labour is largely superseded by the use of machinery, is in use at some works. In this process the agitation of the metal during crystallisation is affected by means of steam. Two pots only are required, one placed above the other; the lower one, which has a capacity of about 36 tons—at least double the upper one—is raised above the floor about 12 inches. The lower pot is covered in at the top with doors and communicates with condensers. A crane is so placed that it can command the two pots and the ingot moulds.

The lead to be desilverised is charged into the top pot by means of the crane; when it has melted the door is removed, and the lead run into the lower pot among the crystals resulting from a previous operation: the crystals melt; the pot is then drossed and a jet of steam introduced. The steam is uniformly distributed by means of a baffle-plate placed above the nozzle. To hasten the formation of crystals, small jets of water are made to play on the surface; the cooled portion sinks into the molten mass, and gradually lowers its temperature to the crystallising point, the action taking place uniformly, owing to the mixing effect of the steam-jet. This part of the operation is carried on until two-thirds of the pot have crystallised; the third remaining liquid is then drained off through two pipes, controlled by valves, the crystals being retained in the pot by means of perforated plates. A 36-ton pot can be worked off in about an hour. After the lead has been tapped out from the working-pot, a fresh charge of lead is run in, having the same richness as the residual crystals from the last operation, and so on.

By the use of this process subsequent softening of the lead is rendered unnecessary, such impurities as antimony,

arsenic, copper, zinc, and iron, being removed by the poling action of the steam; zinc and iron being directly oxidised with liberation of hydrogen. There is admittedly a saving in fuel and labour in this process, and also the cost of softening is avoided. These advantages are, however, about counter-balanced by the original outlay for plant and the expenses for repairs.

The only method which in any way rivals the Pattinson process, and then only for special classes of lead, is that introduced by Parkes. This process depends on the two following facts:—

1. That lead and zinc, however intimately mixed, do not alloy, and on being slowly cooled almost completely separate from one another into two layers.
2. That silver has a greater tendency to alloy itself with zinc than with lead. Consequently, if zinc be melted and well stirred in with lead containing silver, on slowly cooling the mixture the zinc will rise to the surface, carrying with it the silver.

The charge of silver-lead is usually about 15 tons; its temperature has to be raised to a point considerably above that required for Pattinsonising. The quantity of zinc added is regulated by the amount of silver contained in the lead; for lead containing about 50 ozs. of silver to the ton the zinc required would be about $1\frac{1}{2}$ per cent. of the charge. The zinc is added in several portions, the pot being cooled after each addition, causing a crust of argentiferous zinc to form on the surface, which is lifted out of the pot, and, having been drained as much as possible of the more fusible lead, is placed in a smaller pot for subsequent treatment. The lead is finally obtained free from silver, and containing only $\frac{1}{2}$ to $\frac{3}{4}$ of 1 per cent. of zinc, which is sometimes left in, but generally removed by melting the lead in a reverberatory furnace, and exposing its surface to the action of the air until the zinc is all oxidised out, which is readily affected, especially if it be poled with green wood. In some works the zinc is removed by oxidising it with steam:

the advantage of this plan is doubtful. As much lead as possible is allowed to sweat out from the zinc crusts in the pots, after which they are mixed with lime and carbonaceous matter, and placed in retorts or crucibles, where the zinc is distilled off and condensed as far as possible. The residue, containing the silver and some lead, is melted, and the argentiferous lead cupelled for the extraction of the silver. In this process the cost of plant and the expenses for labour are small, the lead can be rapidly treated, and only a small working stock is required. Against this has to be set the cost of the zinc lost, which is considerable.

Uses of Lead.—Many of the uses of lead result from its softness and plasticity, properties which it possesses in a higher degree than any other metal commonly used in the metallic state. The ease with which it may be rolled into sheets recommends it for roofing and for lining sinks, cisterns, &c., particularly since it can be easily adapted to any shape with the aid of a mallet. Again, its softness enables it to be made into pipes, either by casting a very thick cylinder round an iron core and drawing it through progressively diminishing steel dies, or by forcing the melted metal, by hydraulic pressure, through a steel cylinder with a core, from which the solidified metal issues with the required form and dimensions.

The great weight of the metal (specific gravity 11·4) is unfavourable to its employment for roofing, and its ready fusibility (at 620° F.) is another disadvantage, the terrors of a conflagration being sometimes aggravated by the pouring down of the melted lead from the roof.

The poisonous nature of the compounds of lead renders it dangerous to use it for cisterns and pipes with which water, and especially soft water, is to remain in contact for any considerable period, for although lead itself is not acted upon by water, the oxygen of the air, which is always held in solution by water, readily converts a portion of the lead into an oxide of lead, which is dissolved in small quantity by the water; even if a very minute proportion of oxide of

lead be dissolved in the water, repeated doses of it will give rise, in the course of time, to the most painful symptoms. Leaden pipes coated internally with tin, to resist the action of water and air, are made by drawing out two concentric cylinders of lead and tin. The use of lead in connection with cider-vats and presses is highly blameable, for the lead is dissolved in large quantity in contact with the acid liquid, and a moderate draught of such cider may easily contain a poisonous dose of the metal.

The want of tenacity exhibited by lead prevents it from being drawn into thin wire.

Its softness enables lead to mark paper, which rubs off minute particles of the metal ; the pencils in use for metallic memorandum books are composed of lead hardened by the addition of tin and bismuth.

The easy fusibility of lead adapts it to the use of the type-founder, but it is far too soft to be employed alone for this purpose, and is therefore hardened by the addition of antimony.

Type-metal is an alloy of lead with one-third or one-fourth of its weight of antimony. An improved description of type-metal, lately introduced, is composed of two parts of lead, one part of tin, and one of antimony. Another alloy employed for the same purpose contains fifteen parts of lead, one part of tin, and four parts of antimony.

The high specific gravity as well as the fusibility of lead recommend it for making bullets and small shot, where great momentum is required in a small compass. Rifle bullets must be made of very pure soft lead in order that they may easily take the grooves of the rifle, and the iron projectiles of rifled ordnance are coated with lead for a similar reason, but a somewhat harder lead is employed here ; the surface of the shot or shell to be coated is thoroughly cleansed, then dipped into solution of sal-ammoniac, and afterwards into melted zinc, the coating with this metal being found to cause a firmer adhesion of the lead into which the missile is next plunged.

Bullets intended to be discharged from smooth-bore small arms, especially those for breech-loaders, which are of small size, are commonly hardened by the addition of one-fifth of their weight of antimony, in order to give them greater penetration. The bullets employed in shrapnel shells are also composed of four parts of lead and one part of antimony, partly for the sake of penetration, and partly that they may scatter better, bullets of soft lead being liable either to be jammed together by the force of the explosion, or so distorted as to make a very short flight when the shell bursts.

Small shot for fowling-pieces are composed of lead containing from three to six parts of arsenic in a thousand, which has the effect, not only of hardening the lead slightly, but also of enabling it to take a nearly spherical form when the melted metal is dropped through a colander into water. Another condition for securing spherical shot is the proper cooling of the drops before they fall into the water, for if they are suddenly chilled and solidified externally long before the inner portion solidifies, the shrinking of the latter, as it cools, causes the outer layer to collapse, and the shot becomes deformed.

Probably the effect of arsenic in securing the spherical shape is due to its diminishing the contraction of the still liquid lead as it cools after the outer portion has solidified.

In order to cool the drops before they enter the water, they are commonly allowed to fall through the air from a considerable height, either in a shot-tower, or in the disused shaft of a mine; or the same object is sometimes attained by employing a rapid blast of air, when a high fall may be dispensed with. The larger the size of the shot, the more preliminary cooling will they require, so that large shot are allowed to fall through 150 feet, and small shot through 100 feet. In order to prepare the metal, it is usual to alloy a quantity of lead with a large amount of arsenic, and to add this to melted lead in the proper proportion. A ton of soft lead is melted in an iron pot, and 40 lbs. of arsenic added to it; the pot is covered with an iron lid, and the

joints cemented with clay to prevent the arsenical vapour from escaping ; the metal is kept melted for three or four hours, then carefully skimmed, and cast into pigs. The arsenic is added sometimes in the metallic state, sometimes as *white arsenic* (arsenious acid, composed of arsenic and oxygen) or as *orpiment* (sulphuret of arsenic), the two last being decomposed by the lead, and converting a portion of that metal into oxide or sulphuret. Two or three tons of inferior lead having been melted, five or six pigs of the arsenical lead are added, and well stirred up with it ; a small sample is allowed to fall from a height, through a perforated ladle, into water ; if the drops become flattened, too much arsenic has been added, if they are pear-shaped, there is too little. The colanders are wrought-iron bowls about 10 inches wide, perforated with smooth holes varying in size with the description of shot required. In order somewhat to delay the passage of the melted lead through the holes, the colanders are lined with the *cream* or scum of oxide which forms upon the surface of the metal. The temperature of the lead when poured into the colanders is scarcely adequate to scorch straw. The drops then fall from the top of the shot-tower into a vessel of water at the bottom. They are afterwards dried on a hot plate, sifted into different sizes, the deformed shot rejected by gently shaking on a slightly inclined table, when only the spherical shot roll down, and these are polished in a revolving cask containing a little plumbago.

Lead is extensively employed in the construction of vessels for various chemical manufactures, since it resists the action of sulphuric, muriatic, and fluoric acids in a far higher degree than iron, copper, zinc, or tin. Even nitric acid, if strong, scarcely attacks lead, though the diluted acid readily dissolves it. The large chambers in which sulphuric acid is manufactured are built of leaden plates weighing 5 or 6 lbs. per square foot. Here the fusibility of the metal becomes an advantage, for they have to be united by being

burned together, that is, by directing a hydrogen flame along the edges of the plates so as to unite them without the intervention of solder, which would soon be corroded under the action of the acid. This is sometimes called *autogenous soldering*.

Notwithstanding that lead is unacted upon in the cold by strong acids, it is very soon extensively corroded when exposed to the action of air in the presence of carbonic acid, and becomes eventually converted into a mass of *white lead* or (basic) carbonate of lead. Since carbonic acid is produced abundantly by the decay and putrefaction of animal and vegetable matters, metallic lead is much affected when kept in contact with such substances in the presence of air, the oxygen of which unites with the lead to produce an oxide of lead which then combines with the carbonic acid and forms a carbonate. The lead of old coffins is sometimes found to have become almost entirely converted into an earthy-looking mass of white lead in this way, a very thin plate of lead remaining in the centre. The oldest process for the manufacture of white lead depends upon the corrosion of the lead in this manner.

In breech-loading cartridges, where grease is employed as a lubricator, the bullets have sometimes become partly converted into white lead, and have thus increased so much in bulk as to burst open the copper case of the cartridge and render it useless.

Alloys of Lead and Tin.—*Pewter** is an alloy of four parts of tin with one part of lead; it is harder, possesses more tenacity, and melts more easily than either of the metals separately, and provided that the lead does not exceed this proportion, the alloy may be used for drinking-vessels without any danger of lead-poisoning. Since, however, lead is far cheaper than tin, a larger proportion than one-fifth of lead is often employed, when the lead is apt to be dissolved if left in contact with the acetic acid always present in beer.

* Probably corrupted from the French, *potée*, pot. *Potée d'étain* is the French for pewter.

- . Pewter made with the above proportions has the specific gravity 7·8, so that specimens having a higher specific gravity than this will be known to contain more lead.

The solder employed by the pewterer is a very fusible alloy of tin, lead, and bismuth. The solder used for tin-plate is an alloy of lead and tin. *Common solder* contains equal weights of the two metals; *fine solder* contains two parts of tin and one of lead; *coarse solder*, two parts of lead and one of tin. In making solder, the proportions of the metals can be judged of from the appearance of the alloy. When it contains a little more than one-third of its weight of tin, its surface, on cooling, exhibits circular spots due to a partial separation of the metals; but these disappear when the alloy contains two-thirds of its weight of tin. These alloys melt at a much lower temperature than either of their constituent metals. Common solder melts at 385° F.; fine solder at 372° F., whilst the melting-point of tin is 442° F., and that of lead is 620° F.

Soldering is scarcely to be regarded as a merely mechanical adhesion, but depends probably, in part, upon the formation of an alloy between the solder and the surface of the metal to be soldered. Hence it is absolutely necessary that the surfaces to be united by the intervention of solder should be perfectly bright and free from oxide. Several substances are employed to ensure this at the moment of applying the solder; one of the commonest is muriatic (hydrochloric) acid *killed* with zinc, that is, in which a lump of zinc has been dissolved, partly with the object of saturating a portion of the acid, partly to form a chloride of zinc which melts over the surface of the work, dissolving any oxide, and protecting the metal from the oxidising action of the air. Sal-ammoniac (muriate of ammonia), which contains hydrochloric acid combined with ammonia, is also employed, the hydrochloric acid removing any oxide from the metallic surface; sometimes a combination of sal-ammoniac and chloride of zinc is used. Rosin in powder is often sprinkled over the metal to be soldered, when the heat melts it and

forms a varnish to protect its surface from the oxygen of the air.

Hard soldering or *brazing*, for uniting the edges of iron, copper, or brass, is effected with an alloy of brass and zinc made by adding zinc to brass melted in a covered crucible; the alloy is granulated by pouring it through a bundle of twigs held over a tub of water, and before being used for brazing it is mixed with a little moistened borax, which melts when the heat is applied, dissolving off any oxide from the metals, and protecting them from the action of the air. For fine work, a little silver is added to the alloy, which is thus rendered much more liquid when fused.

Terne-plate resembles ordinary tin-plate, but is coated with an alloy of tin and lead; it is largely exported to Canada, where it is employed for roofing.

SILVER.

This metal being, in general, a far less chemically active metal than the preceding, that is, being less likely to enter into and remain in a state of chemical combination with other substances, is much more frequently met with in the metallic or native state.

Native silver has generally the appearance of metallic twigs and branches, which are sometimes composed of crystals of silver strung together. The silver-mines of Potosi exhibit such specimens. Native silver is also found at Kongsberg in Norway, at Andreasberg in the Hartz, Freiberg in Saxony, and Schemnitz in Hungary.

The native metal generally contains small quantities of gold and copper. At Kongsberg a yellow alloy is found which contains silver with more than one-fifth of its weight of gold. An amalgam of silver with mercury is found in large quantity in the silver-mines of Coquimbo, Chili.

Sulphuret of silver, or *silver-glance*, containing, in its pure state, 87 parts of silver combined with 13 of sulphur, is one of the commonest forms of combination in which silver occurs in nature. It has been found, in a pure state, in Cornwall, Norway, Hungary, Saxony, Bohemia, Mexico, Peru and the United States. It has a slight lustre, and a dark grey colour, possessing also, which is remarkable in an ore of this description, a good deal of malleability and flexibility, and it is so soft that it may be cut with a knife. It is also distinguished by its fusibility, being easily melted even in an ordinary flame.

The sulphuret of silver is more abundant in association with the sulphurets of other metals; thus, in argentiferous galena, with sulphuret of lead; in grey copper ore, with the sulphurets of copper, antimony, arsenic, iron, and zinc; in *brittle silver ore*, with the sulphurets of antimony, iron, and copper; in *red silver ore*, with sulphuret of antimony or sulphuret of arsenic. Blende, iron pyrites, mispickel, and some other minerals sometimes contain a minute proportion of silver, which may be extracted with profit, incidentally to other processes.

Horn-silver or *chloride of silver* contains, when pure, 75 parts of silver united with 25 parts of chlorine. Good specimens of this ore exhibit, as its name implies, some resemblance to horn in appearance and softness. It is found abundantly in Chili and Peru, sometimes in large fragments, but more commonly in very small cubical crystals disseminated in a ferruginous rock.

The *butter-milk ore* of the German miners contains chloride of silver mixed with a large proportion of clay.

Chloride of silver has been found, in small quantity, in Cornwall. *Bromide* and *iodide of silver* are also found in Mexico and Chili.

In consequence of the high price of silver, it admits of being extracted with profit even from ores which contain a very small proportion of the metal, especially if some other useful metal can be extracted at the same time. Thus,

silver may be profitably obtained from galena containing only two parts of silver in a thousand, and even a smaller quantity than this is extracted from some copper ores. In these cases, the lead and copper respectively are extracted by the ordinary smelting processes, and are then subjected to special treatment for the extraction of the silver. It may be well to describe the principal methods in use for this purpose before considering the metallurgic treatment of the ores of silver with the sole object of extracting that metal.

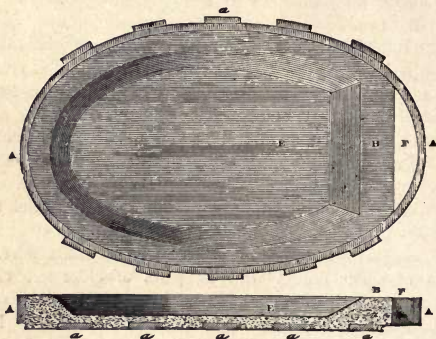


FIG. 99.—English Cupel or Test.

Extraction of Silver from Lead by Cupellation.—Most of the silver produced in this country is extracted by this process from the rich lead obtained in Pattinson's desilverising process (p. 320).

The process of cupellation, which is one of the most attractive metallurgic operations, derives its name either from the German *kuppel*, a *cupola* or *dome*, in allusion to the shape of the German cupellation furnace, or from a diminutive derived from the Latin *cupa*, a cup, referring to the concave hearth upon which the process is carried out.

The extraction of silver from lead by cupellation depends upon the facility with which the latter metal is converted into an oxide by the action of air at a high temperature, whilst silver is almost entirely unaffected; the oxide of lead being easily melted, is partly removed from the surface in a liquid state, and partly absorbed by the porous hearth upon which the silver remains. In England, this hearth, which is

called the *cupel* or *test*, is an oval frame of wrought iron (A, Fig. 99), 5 feet long and $2\frac{1}{2}$ feet wide, which is crossed by five iron bars (a) $3\frac{3}{4}$ inches in breadth. This frame is filled with finely-powdered bone ashes moistened with water, in which a little pearl-ash (carbonate of potash) has been dissolved; this is well consolidated by beating, and scooped out until it is about $\frac{3}{4}$ of an inch thick over the cross bars, leaving a flat rim of bone-

ash all round, about 2 inches wide, except at one end (B), the front or *breast* of the cupel, where it is 5 inches wide; through this a channel (F) is cut, to allow the melted oxide of lead to flow off without coming in contact with the iron, which it would corrode very seriously. This cupel rests upon a car, in order that it may be wheeled into its place under the reverberatory furnace (Fig. 100), of which

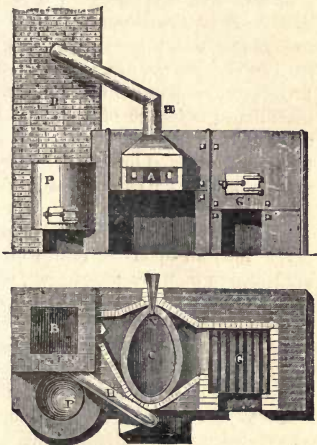


FIG. 100.—English Furnace for the cupellation of Lead.

it forms the hearth (c), where it is arranged so that the flame of a coal fire (g) passes directly across it to the two flues, which run into a chimney (b), about 40 feet high. A blast pipe or tuyère (n) enters the cupel at the end opposite to that through which the oxide of lead escapes, and throws a blast of air over the surface of the metal at the rate of about 200 cubic feet per minute. Opposite to the tuyère is a hood with a pipe (h) for carrying the fumes into the chimney. The cupel is very gradually heated nearly to redness, and almost filled with the lead to be treated, which is ladled in from an iron pot (p), in which it has been previously melted. From 500 to 600 lbs. of lead are introduced at once. In a short time, the surface of the metal becomes covered with oxide of lead, or *litharge*,* in a melted state; the blast is then turned on, and drives the litharge in wavelets off the surface, through the channel made for its escape, into a cast-iron pot outside the furnace. When this channel is very much corroded, it is closed, and another is cut. In proportion as the lead upon the hearth diminishes, fresh portions are ladled in from the melting-pot, so as to keep the lead at about the same level. The process is continued for sixteen or eighteen hours, in which time four or five tons of lead will have been added, and an alloy containing about eight parts of silver in a hundred will be left in the cupel. A hole is then made in the bottom, through which the metal is run out and cast into pigs. A fresh charge is introduced into the cupel, and the operation continued; one cupel will often last for forty-eight hours, and is capable of treating 5 cwts. of lead per hour, with a consumption of 1 cwt. of coal. When about 3 tons of the rich alloy (containing 8 per cent. of silver) have been obtained, it is again subjected to cupellation in the same furnace, but in a cupel which has a concavity at the bottom intended for the reception of the cake of silver, which will weigh about 500 lbs. This second operation is

* From two Greek words, signifying *stone* and *silver*.

necessary, because the litharge which is formed from this very rich alloy contains a considerable proportion of silver, so that the lead obtained by smelting it (p. 340) and by smelting the cupel, which absorbs a large proportion of litharge, contains 30 or 40 ozs. of silver in the ton, and must be treated for that metal.

The appearance of the metal in the cupel as the last portions of lead are removed in the form of litharge and absorbed into the bone-ash, is very beautiful. When the film of oxide becomes so thin that the bright silver beneath can reflect the light through it, a decomposition of the light into its constituent colours takes place, and the most brilliant rainbow tints are seen upon the metal, their beauty being enhanced by the rapid rotation of the film. As soon as this film of oxide has been absorbed by the cupel, the splendid surface of the melted silver shines out (*fulguration, coruscation, or brightening*).

During the cooling of the cake of silver, some very remarkable phenomena are observed. When a thin crust of metal has formed upon the surface, the silver beneath it assumes the appearance of boiling, and the crust is forced up into hollow cones about an inch high, through which the melted silver is thrown out with explosive violence, some of it being splashed against the arch of the furnace, and some solidifying into most fantastic tree-like forms several inches in height. This behaviour of silver has been shown to be due to its property of absorbing mechanically (*occluding*) oxygen, at a temperature above its melting-point, which it gives off as it approaches the point of solidification, the escaping gas forcing up the crust of solid silver formed upon the surface.

A considerable proportion of lead and silver is carried off by the blast, in the form of vapour, and is partially recovered, as oxide, from the flues of the furnace.

In some cupellation furnaces, instead of employing a blowing machine, a current of air is directed over the surface

by means of a jet of steam issuing from a tube surrounded by a wider one through which the air is dragged by the mechanical action of the steam. This is said to hasten the operation, and to produce litharge of better quality.

The litharge produced in the English cupellation furnace is reduced to the metallic state in a reverberatory furnace with a hearth measuring 8 feet by 7, which is lined with bituminous coal; this soon becomes converted into a porous coke, which protects the clay hearth of the furnace from

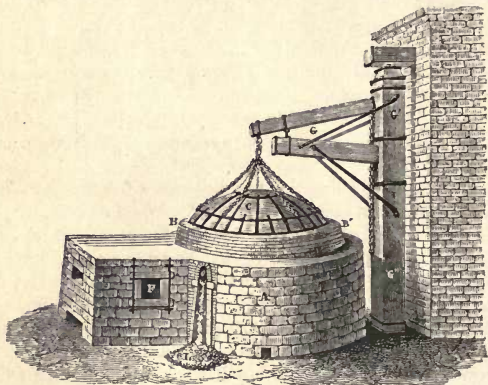


FIG. 101.—German Cupellation-furnace.

being corroded by the melted litharge, and forms a filter through which the lead runs towards the opening from which it is tapped. About 3 tons of litharge mixed with 6 cwts. of small coal are charged at once; the carbon of the coal removes the oxygen from the oxide of lead composing the litharge, and reduces the lead to the metallic state. The lead thus obtained usually contains 30 or 40 ozs. of silver in a ton, and is introduced into its appropriate place in a series of Pattinson's pans (p. 320).

The *German cupellation-furnace* (Figs. 101, 102) differs from the English in having a fixed instead of a moveable hearth (A), covered with an iron dome (C) lined with clay, which is capable of being lifted off by a crane (G). The hearth is circular, about 10 feet wide, and is lined with marl, or with an intimate mixture of clay and lime well beaten and hollowed out like a saucer, with a circular cavity about 20 inches wide and $\frac{1}{2}$ inch deep, in the middle, for collecting the cake of silver. Wood ashes, previously washed and well beaten down, are sometimes employed instead of marl. Two tuyères (a) direct the blast across the hearth, and are pro-

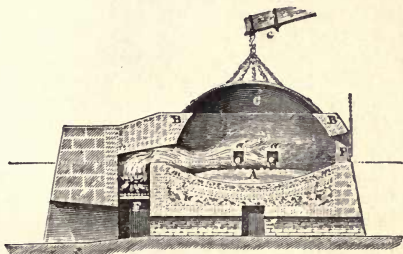


FIG. 102.—Section of German Cupellation-furnace.

vided with *butterfly-valves* for guiding the blast over the surface of the metal. The fire-place (F) is situated in a square furnace adjoining the hearth, and is supplied, when practicable, with wood, which gives a longer and clearer flame than coal. Impure leads can be treated by the German process, whilst the English method of cupellation is adapted for those which are comparatively free from antimony and copper.

From 4 to 17 tons of lead can be cupelled at once, according to the size of the hearth, the pigs being placed upon a thin layer of straw. The heat is gradually raised so as to melt the lead, no blast being employed for the first three hours. When the metal is in a state of tranquil fusion, the

surface is skimmed to remove the dross, and the bellows are worked at the rate of about four or five strokes a minute, in order to renew the air over the surface, so as to promote oxidation. In about two hours, a stronger fire is applied, and the crust of oxide and of various mechanical impurities is skimmed off the surface through the opening (*o*) provided for the escape of the litharge. About an hour and a half is occupied in thoroughly cleansing the surface of the lead.

The blast is now freely directed upon the melted metal, so as to produce litharge abundantly, and to drive it in waves through the outlet (*o*), which is deepened by the workman in proportion as the level of the lead falls. The litharge flows out on to the floor of the shop as at *L*. After continuing the process for a period varying between seven hours and sixty hours according to the amount of lead, the removal of the lead is complete so far as it can be effected in this furnace, and the phenomena described at p. 339 as indicating the termination, are witnessed. A wooden spout is then introduced into the charging-door, through which water is carefully poured upon the surface of the silver to solidify it into a cake, which undergoes a subsequent refining to complete its purification.

The first portions of oxide which form upon the surface in this process contain, beside the oxide of lead, oxides of antimony, iron and other impurities, and yield, when reduced, a very impure lead, fit only for making shot or type-metal. The last portions of litharge, amounting to about $\frac{1}{8}$ th of the whole, are set aside to be reduced separately, when they furnish a lead rich in silver. When bismuth is present in the lead, the last portions of litharge have a green colour. A certain quantity of the intermediate portion of litharge (containing less than half an ounce of silver in the ton) is sent into the market as such, being useful to the manufacturers of glass and earthenware, of sugar of lead, &c. In the market, it finds a readier sale when in reddish brown scales or flakes, which are produced by running it out, when

melted, into large iron vessels, and allowing it to cool in a draught of air. When these vessels are inverted, the mass of litharge is easily turned out and broken into a flaky powder. Impure litharge cannot be converted into red litharge, so that the colour is evidence of its purity.

The cupel, which is largely impregnated with litharge and contains some silver, is broken up and smelted in order to extract those metals.

At Andreasberg and Freiberg, the silver ores are melted with the lead (already containing some silver) upon the cupel itself, when any sulphuret of silver which they contain is decomposed, its sulphur combining with oxygen to form sulphurous acid, and the silver being dissolved by the lead.

At Kongsberg, the use of a blast heated to about 400° F. has been attended with a great saving of time and fuel, with the additional advantage of cupelling leads containing ten or twelve parts of copper in the hundred, which are too difficult to fuse on the cold blast cupel. In a furnace six feet in diameter, a charge of $3\frac{1}{2}$ tons of lead is cupelled in seven hours.

The silver obtained by cupellation is liable to contain small quantities of lead, bismuth, antimony, copper and gold, the three first of which render it brittle. It is therefore generally subjected to a refining process, which consists in exposing it in a melted state to the action of the air, when the foreign metals, with the exception of the gold, are oxidised and absorbed by the test. The operation is performed either in an ordinary cupellation furnace, or in another constructed upon the same principle.

Extraction of Silver from the Ores of Copper.—The ores of copper containing silver are smelted chiefly at Mansfeld, the copper being extracted in the form of *black copper* by the process described at p. 245. From the black copper the silver is extracted by the process of *eliquation*, which consists in alloying it with a large proportion of lead, and afterwards

melting out the latter metal by a moderate heat, when it carries all the silver with it, to be afterwards extracted by cupellation. The black copper is broken into small fragments, or granulated by melting it and running it into

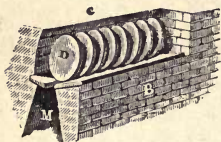


FIG. 103.—Liquation-hearth.

water, and is then fused in a small blast furnace with from two to four times its weight of lead, that which already contains silver being chosen if possible, and the two metals thrown in alternately. The alloy of lead and copper is

cooled quickly in thick cast-iron moulds, and chilled with water in order to avoid the separation of the two metals, being thus cast into round cakes 18 inches in diameter and 3 inches thick. These cakes are placed on a *liquation-hearth* (Fig. 103), which is a gutter made by two sloping cast-iron plates with a space between them through which the lead trickles down. The cakes of copper-lead (D) are set on

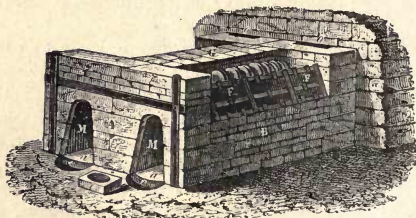


FIG. 104.—Liquation-hearth.

edge across this gutter, with pieces of wood to keep them apart; the gutter is then shut in with iron plates (F, Fig. 104) and filled with charcoal. A wood fire being made in the space (M) beneath the gutter, the charcoal takes fire and melts the lead, which runs into a receptacle (O) outside the

furnace, carrying the silver with it. This lead is cast into ingots from which the silver is extracted by cupellation. The liquation occupies three or four hours. The copper cakes still retain about one-fourth of lead and some silver, which are extracted by exposing them to a higher temperature in a *sweating-furnace* (Fig. 105) where they are placed over a number of fire-brick channels (F) in which a wood fire is made. The bulk of the lead is converted into oxide by the air in the furnace, so that a quantity of litharge containing oxide of copper and silver collects at the bottom of the channels, and is fused with the black copper in the blast furnace employed for preparing the cakes to be submitted to eliquation.*

Extraction of Silver from the Ore by Melting with Lead.—At Kongsberg in Norway, where the ore contains its silver in the metallic state,

it is extracted by simply melting the dressed ore with its own weight of lead, when an alloy containing about one-third of its weight of silver is obtained, which is submitted to the process of cupellation. When ores containing sulphuret of silver are melted with lead, this metal removes the sulphur in the form of sulphuret of lead, and the liberated silver is dissolved by another portion of lead.

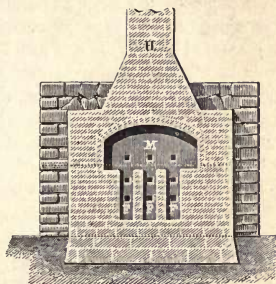


FIG. 105.—Sweating-furnace. M, Back-wall with flues communicating with the chimney H.

AMALGAMATION PROCESS FOR THE EXTRACTION OF SILVER FROM ITS ORES.

The process of amalgamation is so called because the silver is extracted in the form of an *amalgam* with mercury,

* This process is now generally replaced by a *wet* method.

and there are two modes of carrying it out, the choice being regulated by the scarcity or abundance of fuel in the locality. Thus, the amalgamation process employed in Mexico and Chili, where fuel is dear, is very different from that in use in Freiberg where it may be had in abundance.

Mexican Process of Amalgamation.—In Mexico and Chili, where the silver ores obtained from the western slopes of the Cordilleras are treated, the thorough pounding of the ores and of the materials added to them, by the comparatively inexpensive agency of water-power, horses and mules, is employed to facilitate the chemical changes which would be promoted by the action of fire, if fuel were more abundant. Time is also an important element in this process, which consists essentially in converting the whole of the silver into chloride of silver, to be afterwards brought to the metallic state by the action of mercury, and dissolved by the latter, in the form of an amalgam.

The ore sometimes contains only 35 ozs. of silver in the ton, partly in the metallic state, and partly as chloride and sulphuret of silver, and is carefully picked over in order that the worthless portions may be rejected. It is then crushed in oblong mortars under wooden pestles, shod with iron, weighing about 200 lbs. each, and raised by cams projecting from an axle moved by a water-wheel. The crushed ore is ground with water into a mud, under granite stones, which are made to revolve in a granite bed by the labour of mules. This mud is transferred to the *amalgamation-floor*, an enclosure, about 300 feet by 240 feet, paved with stone. It is there mixed, with wooden shovels, with a proportion of common salt, varying from one to five parts for every hundred parts of ore, and, in order to effect a thorough intermixture, a number of horses are made to trample the mud in the enclosure, after which it is left at rest for some hours.

On the following morning, the horses are again turned into the amalgamation-floor for an hour, after which, copper pyrites which has been roasted and ground to powder (when

it is called *magistral*) is added in the proportion of $\frac{1}{50}$ th or $\frac{1}{100}$ th of the weight of the ore. The mud is then trampled for five or six hours by horses.

The mixture is now ready for the amalgamation with mercury, which is sprinkled upon it from a bag of coarse canvas, in quantity about twice that of the silver present in the ore. It is then trodden again by horses, and well turned with wooden shovels, these operations being repeated every other day until, on washing a small sample in a bowl, it is found that all the mercury which has been added is in combination with the silver, as an amalgam, no globules of mercury being visible.

The following appear to be the principal reactions upon which this process depends : double decomposition between sulphate of copper and chloride of sodium, producing chloride of copper (cupric chloride) ; double decomposition between chloride of sodium and sulphide of silver, producing chloride of silver ; reduction of chloride of silver by mercury, forming a heavy amalgam and chloride of mercury. A certain amount of cuprous chloride is also formed, which reacts with sulphide of silver, as in the case of cupric chloride. This secondary reaction takes place at the moment double decomposition is occurring between the cupric chloride and the sulphide of silver, and is brought about by the copper of the chloride concerned in the reaction reducing to a lower chloride a further quantity of cupric chloride, in preference to combining with the sulphur of the sulphide of silver, so that, in the result, chloride of silver, cuprous chloride, and free sulphur are produced. This secondary reaction can, however, only take place to a limited extent.*

The examination of the amalgam shows whether too much sulphate of copper (in the form of *magistral*) has been added. If this be the case, a quantity of the mercury will be found to be so finely divided as to form a dark-coloured

* For further details see investigations by A. K. Huntington, *Jour. Soc. Chem. Industry*, July 1882.

mud, and occasional brown spots of metallic copper will be visible. A great loss of mercury would ensue if the magistral had been added in excess, because the chloride of copper formed from it would be decomposed by the mercury, yielding a chloride of mercury (calomel) and subchloride of copper.

If it is found that this error has been committed, a little lime is added to the contents of the amalgamation-floor to decompose the chloride of copper (forming chloride of calcium and oxide of copper) and prevent any further waste of mercury.

It sometimes occurs, on washing the sample, that globules of mercury are perceived, showing that it has not entirely united with the silver. This is due to an insufficient supply of magistral, in consequence of which the amount of chloride of copper formed has not been sufficient to convert the whole of the silver into chloride of silver, a form in which it is much more readily acted upon by the mercury than when it is in the state of sulphuret of silver, because the chloride of silver is capable of being dissolved by the strong solution of common salt existing in the mud, and is thus presented to the mercury in the condition most favourable to chemical action. If necessary, a further addition of the roasted copper pyrites is made before proceeding with the amalgamation.

After about a fortnight, a fresh quantity of mercury is added, rather less than one-third of the first addition, and the operation of trampling is repeated. When this mercury has also been taken up, a third portion, about half as large again as the second, is added, in order to dissolve the amalgam of silver in an excess of mercury. After a thorough incorporation by trampling, the mixture is at once shovelled into barrows and taken to the washing-vat, which is a circular cistern 8 feet wide and 9 feet deep, in which the mud from the amalgamating-floor is well stirred up with water, by an agitator worked by four mules, fresh water constantly running

into the cistern, and carrying off the earthy matter over the side.

The liquid amalgam left behind is thrown into a leather bag with a canvas bottom, through which the excess of mercury is strained off, leaving a pasty amalgam containing about $\frac{1}{3}$ th or $\frac{1}{6}$ th of its weight of silver, which is subjected to pressure in order to squeeze out more mercury and convert it into a hard solid mass. This is moulded into wedge-shaped masses of about 30 lbs. each, which are built up into a circular tower on a copper stand having a hole in its centre through which a pipe passes in order to conduct the mercurial vapours into a tank of water placed beneath. The pile of amalgam is covered with an iron bell, the opening of which fits tightly upon the copper stand, any crevices being carefully filled up with a cement. A temporary furnace is built with bricks around the bell, and kept full of burning charcoal for about twenty hours, when all the mercury is converted into vapour, which condenses in the water beneath, and the silver is left as a hard mass; this is broken up, melted, and cast into bars weighing about 1000 ozs. each.

The period occupied by the amalgamation process varies greatly according to the nature of the ores, extending sometimes over little more than a fortnight, and sometimes requiring six or eight weeks.

The loss of mercury in the process is very considerable, amounting to about 24 ozs. for every pound of silver obtained, being due, in great part, to the *flouring* or fine division of the metal, which is then carried away in the process of washing, together with all the silver which has been dissolved in it.

The *hot amalgamation*, applied for extracting the silver from rich ores which contain it either in the metallic state or as chloride, bromide or iodide, only occupies five or six hours, and entails less loss of mercury. It consists in boiling

the finely-powdered ore with water, common salt (10 or 15 parts for a hundred of ore), and mercury, in pans with copper bottoms, when the chloride, bromide, and iodide of silver are decomposed by the copper, forming a chloride, bromide, or iodide of copper, and liberating the silver, which is dissolved by the mercury.

The extraction of silver (and of gold) by amalgamation is very much facilitated by adding a minute proportion of *sodium* to the mercury employed, for it is found that mercury so treated attacks and dissolves silver and gold much more readily than pure mercury, and that it is very much less liable to assume that finely divided state in which it is so readily washed away and lost.

Silver ores containing complex sulphides, galena, blende, &c., cannot be successfully treated by the ordinary patio process: only a small portion of the silver is extracted. Such ores have recently been satisfactorily dealt with in Chili, by a process invented by Krölinke. He employs cuprous chloride, prepared by heating a solution of sulphate of copper and common salt with metallic copper. The amalgamation is effected with zinc and mercury, in closed barrels, in the same way as the Freiberg amalgamation process presently to be described. The process is complete in a few hours.

Another process, which has been introduced to deal with similar kinds of ores in Nevada, Colorado, California, &c., is known as the *Washoe*. In this process the ores are ground with water in iron pans heated by steam, and mercury is added. The sulphide of silver is decomposed by the iron, sulphide of iron being formed, and metallic silver set free, which is dissolved by the mercury.

Amalgamation Process for Extraction of Silver at Freiberg.—The process employed, until recently, in Saxony for extracting silver is far less wasteful of mercury than the Mexican process, the loss varying from 4 to 12 ozs. per lb. of silver, since this valuable metal is employed

only for the purpose of dissolving the silver after it has been brought into the metallic state, whereas in Mexico, a part of the mercury is used in removing the chlorine from the chloride of silver in order to liberate the metal. In the Saxon process, as in the Mexican, the whole of the silver is converted into chloride of silver, from which the chlorine is removed by the action of metallic iron, and the silver, which has thus been reduced to the metallic state, is dissolved by mercury.

The ore treated at Freiberg contains the silver chiefly as a sulphuret, but it also contains the sulphurets of several other metals, particularly of antimony, bismuth, arsenic, iron, copper, lead and zinc. The different kinds of ore are sorted and mixed so that the mixture may contain less than one part of copper and five parts of lead in the hundred parts, for both these are easily taken up by the mercury, causing undue consumption of that metal. The silver present in the mixed ores should amount to about 80 ozs. in the ton. The presence of a large proportion of iron pyrites (bisulphuret of iron) is necessary for the subsequent chemical changes, so that the mixture is made to contain about one-third of its weight of that mineral.

The ore thus prepared is ground to a coarse powder, mixed with one-tenth of its weight of common salt, and roasted upon the hearth of a reverberatory furnace at a dull red heat, for about two hours, care being taken to turn it over frequently, and to avoid fusion. During the roasting, the oxygen of the air converts the arsenic and antimony into their respective oxides, which pass off as a thick white smoke, together with some sulphurous acid produced by the combustion of the sulphur. The sulphurets of copper, iron and silver, also combine with oxygen, and become converted into the sulphates of those three metals.

The temperature is then raised to enable the sulphates of copper, iron and silver to decompose the chloride of sodium

(common salt) yielding sulphate of sodium, and chlorides of copper, iron and silver. Other chemical changes also occur in this stage, but it will not be necessary to trace them, since they do not affect the extraction of the silver.



FIG. 106.—Amalgamation Cask. *a*, Opening for charging and discharging. *r r'*, Toothed wheel for receiving motion from the axis (Fig. 82).

The first roasting is effected with a coal fire, and the second heating with fir-wood.

The deep brown roasted ore, containing the sulphate of silver, is now sifted, and the lumps mixed with more salt, and roasted again in order to complete the change.

The sifted powder is ground to a very fine meal, and introduced in charges of half a ton into strong oaken casks (Fig. 106),

about three feet each way, containing three hundred weights of water, and mounted, to the number of twenty, on cast-iron

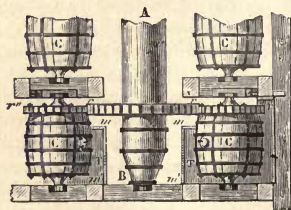


FIG. 107.—Amalgamation of Silver Ores at Freiberg. *r r'*, Toothed wheels for transmitting motion from the axis *A B*. *m m'*, Troughs for receiving the contents of the casks *C*.

axes, so that they can be made to revolve at pleasure by being connected with the axle (*A*, Fig. 107) of a water-wheel. About one hundred weight of wrought iron, in fragments, is put into each of the casks, which are then made to revolve at the rate of ten or twelve turns in a minute. The

iron removes the chlorine from the chloride of silver, producing chloride of iron and metallic silver, which remains dispersed through the mixture in a finely-divided condition.

The iron also decomposes the chloride of copper in a similar manner, yielding finely divided metallic copper.

After having revolved for about two hours, the casks are opened for the purpose of introducing the mercury to dissolve the silver ; but before adding this, the contents must be brought to a proper consistence ; if they are too thick, they could not be well mixed with the mercury, and some more water must be added ; on the other hand, if they are very liquid, the mercury will at once go to the bottom, and will never be thoroughly distributed through the mixture ; some more of the prepared ore must then be introduced. 5 cwts. of mercury are then poured into each of the casks, and they are made to revolve, at the rate of about twenty-five turns in a minute, for sixteen or eighteen hours, when the mercury dissolves the metallic silver and copper. During this process, the contents of the casks are twice examined in order to see if they have a proper consistence. The chemical action which takes place in the casks raises their temperature, even in winter, to about 100° F.

In a more modern arrangement, the casks are fixed in a vertical position, the materials being mixed by a revolving agitator with iron arms. Steam is admitted into the casks through holes at the bottom, in order to facilitate the amalgamation by its heat.

When this rotation is finished, the amalgam of silver and copper is found interspersed in minute globules throughout the mass ; in order to collect it, the casks are filled with water, and revolved eight times in a minute, during about two hours. They are then turned with their wooden bungs (*a*, Fig. 108) downwards, and the pegs stopping the small openings in the bungs being withdrawn, a tube with a stopcock is inserted ; so that the amalgam may run out into a trough beneath. The mixture in the casks is emptied out through a grating which retains the pieces of iron, whilst the mud is collected in tubs, where it is stirred with water and allowed to deposit the small quantity of amalgam which has been carried away.

The mud is afterwards allowed to settle down so that it may be again treated by the amalgamation process, when it yields about $4\frac{1}{2}$ ounces of silver for a ton.

The liquid amalgam of silver and copper is strained through canvas bags, as in the Mexican process, when a quantity of liquid mercury runs off (containing about 20 ozs. of silver to the ton), leaving in the bags a pasty amalgam containing about 30 parts of mercury, 4 parts of silver, and

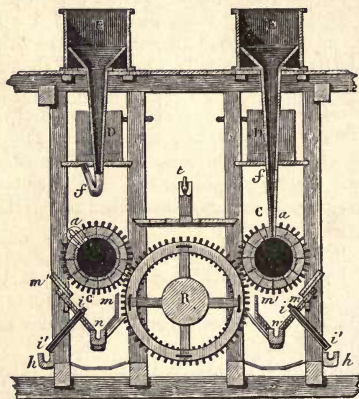


FIG. 108.—Amalgamation of Silver Ores at Freiberg. *E*, Reservoirs for the ground mineral. *f*, Leathern pipe for conveying it into the opening *a* of the cask *C*. *D*, Cisterns containing water. *m n m'*, Receptacles for the amalgam and spent charge. *i i'*, Pipe for conveying the amalgam into the gutter *h*.

1 part of copper, with small quantities of other metals which existed in the ore, particularly lead, bismuth, zinc, antimony, and gold. The amalgam is sometimes allowed to settle in a narrow wooden cylinder, 8 feet high, before being strained in the bags, when it separates into two layers, the lower consisting chiefly of mercury, which need not be strained.

The pressure is sometimes applied to the amalgam by a

screw, and sometimes by the hydraulic press; in the latter case the amalgam is placed in an iron cylinder with a wooden bottom through which the liquid mercury is pressed out.

This pasty amalgam is made up into balls which are placed in iron dishes (Fig. 109) supported at about five inches apart by an iron rod which runs through their centres, and stands, upon four feet, in an iron basin of water. When about 3 cwts. of amalgam have been placed in the dishes, an iron bell is let down over them, by a crane, its lower opening resting in the water. A fire is then made around the upper part of the bell, first with wood, then with turf, and finally with charcoal. In this way the bell is made red hot, and the mercury is converted into vapour, condensing in the water beneath, which is kept constantly cool by the passage of cold water through a wooden trough in which the iron

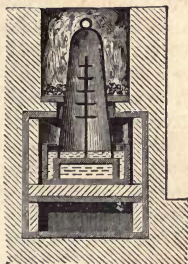


FIG. 109.—Distillation of the Amalgam of Silver.

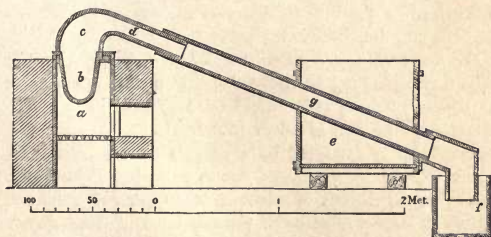


FIG. 110.—Apparatus for distillation of the Amalgam of Silver.

basin stands. After about eight hours, no more globules of mercury are heard to drop into the water, and the distillation is finished. On removing the bell, spongy masses of

the alloy of silver, copper, &c., are found in the iron dishes.

A more modern and economical apparatus for the distillation of the pasty amalgam consists of an iron crucible (*b*, Fig. 110) 22 inches wide and 11 inches deep, which is heated by a charcoal fire (*a*), the vapour of mercury being conducted by an iron hood (*c*) into an iron condensing tube (*g*) which traverses a cistern of water (*e*). The inside of the crucible is coated with lime, and an iron plate coated with lime is fitted into the crucible, which is withdrawn with the silver adhering to it at the close of the operation. 4 cwts. of amalgam are distilled in five hours.

The alloy left after distilling the amalgam is melted in crucibles made of a mixture of fire-clay and plumbago, and briskly stirred with an iron rod. Fumes of the oxides of bismuth, zinc and antimony are given off, and a scum containing oxides of lead and copper forms upon the surface, and is skimmed off. When no more dross appears upon the surface, the metal is cast into ingots. It still contains about one-seventh of its weight of copper, from which it is purified by melting it with lead, and subjecting it to cupellation (p. 336), when the litharge dissolves the copper in the form of oxide, and the silver is left pure.

Treatment of Copper-matts for Silver.—When the last matts (corresponding to the Welsh fine metal) obtained in the copper-smelting at Mansfeld (p. 245) contain any considerable proportion of silver (not less than $2\frac{1}{2}$ parts in a thousand), it is extracted by a process closely resembling that employed at Freiberg. Since, however, these matts consist almost entirely of sulphuret of copper and sulphuret of iron, so large a quantity of the sulphates of these metals is formed during the roasting, that it is found necessary to mix the roasted ore with a quantity of chalk equal to that of the common salt; when the lime in the chalk decomposes the sulphates of iron and copper, forming sulphate of lime, and leaving the iron and copper in the form of oxides.

The mud emptied from the tubs at the close of the amalgamation process, at Mansfeld, is mixed with clay, and made up into cakes, which are dried and smelted in a small blast-furnace to obtain the copper.

Processes employed to supersede the Amalgamation of Silver Ores. (*Augustin's Process.*)—Several methods have been proposed from time to time to avoid the use of mercury in extracting silver from its ores. For example, after roasting the ores, first by themselves, and afterwards in admixture with common salt, as in the Freiberg process, to convert the whole of the silver into chloride, the latter is dissolved out with a saturated solution of common salt, and the chloride of silver in the solution is decomposed by leaving it in contact with scraps of copper, when the latter combines with the chlorine, forming a chloride of copper, and the silver is separated in the finely-divided metallic state, to be afterwards melted and cast into ingots. It has been recommended to mix chlorine-water with the solution of common salt employed in this process, in order to extract the gold at the same time.

This process has been employed with satisfactory results, at Freiberg, for the extraction of silver from the matt containing two-thirds of its weight of copper obtained in the copper-smelting process. The matt, having been roasted by itself, and afterwards with common salt, is stamped to powder, sifted, and placed, in quantities of about 6 cwts., in tubs with perforated bottoms, over which a linen strainer is stretched. A hot strong solution of salt being let in from a reservoir, dissolves the chloride of silver and carries it through the strainer into tubs also provided with a strainer, and containing some spongy copper (cement copper, see p. 248), which is gradually dissolved, as chloride of copper, whilst metallic silver is deposited in its stead; this is removed from the tubs, washed with muriatic acid to remove particles of copper, then with water, and moulded into small balls, which are dried and fused.

The liquid containing the chloride of copper and common salt is conveyed into another series of tubs containing copper, in order to ensure the complete removal of the silver, and afterwards into vessels containing iron, which causes the separation of the metallic copper, to be used over again for precipitating the silver.

The matt, having been washed with solution of salt, until a copper plate dipped into the liquor is no longer whitened by the deposition of silver, is washed with water, and taken to the copper-smelting furnace (p. 245).

Patera's process, as carried out at Joachimstal, is applicable only to rich ores, and consists in roasting with common salt, as at Freiberg, in order to convert the silver into chloride of silver, which is then dissolved out by a cold dilute solution of *hyposulphite of soda*, which takes up chloride of silver much more readily than common salt does. The solution is then mixed with a solution of *sulphuret of sodium*, which produces a black precipitate of sulphuret of silver, and leaves the solution of hyposulphite ready to be employed for treating a fresh portion of ore. The sulphuret of silver is collected on a canvas strainer, washed, dried, roasted to burn off part of the sulphur, and melted in black lead crucibles with metallic iron, which takes up the rest of the sulphur, in the form of sulphuret of iron, and leaves metallic silver. The sulphuret of iron dissolves a part of the silver, and is worked up with a fresh charge of ore.

Ziervogel's Process.—A still simpler process consists in roasting the ore or matt containing the sulphurets of copper, iron and silver, so that these may combine with oxygen from the air, and become converted into sulphates, which are then heated so strongly as to decompose the sulphates of copper and iron, leaving these metals as insoluble oxides, whilst the sulphate of silver is left undecomposed, and may be dissolved out by water and placed in contact with metallic copper, which separates the silver in the metallic

state, and takes its place in the solution, yielding sulphate of copper.

Ziervogel's process has been attended with very satisfactory results in its application to the extraction of silver from the copper-matts at Mansfeld, containing, in 100 parts, 80 parts of subsulphuret of copper, 11 parts of sulphuret of iron, and $\frac{2}{3}$ th part of sulphuret of silver. The loss of silver experienced in treating this matt by the amalgamation process amounted to nearly $\frac{1}{10}$ th of the metal present, by Augustin's process to nearly $\frac{1}{12}$ th, and by Ziervogel's process to only $\frac{1}{120}$ th.

The matt, having been granulated, ground, and sifted, is roasted with great care and judgment, in reverberatory furnaces, seven of which are connected with a single chimney 154 feet high. The roasting occupies about ten hours, and is continued until a small sample taken out and mixed with water gives a liquor which produces a strong precipitate of chloride of silver on the addition of common salt (chloride of sodium), and has a light blue colour, from a little sulphate of copper having been left undecomposed.

The roasted mass is then introduced, in quantities of 5 cwts., into tubs similar to those used in Augustin's process (p. 357), and treated with hot water containing a little sulphuric acid, until the liquor which runs off no longer becomes milky when mixed with common salt, showing it to be free from silver. The solution of sulphate of silver, thus obtained, is run into tubs containing copper, where the silver is precipitated, and is afterwards washed with diluted sulphuric acid to remove adhering copper.

The sulphate of copper in solution is decomposed by metallic iron, and the copper is employed for precipitating the silver.

The copper matt of Freiberg is sometimes roasted, to convert the sulphuret of copper into oxide, and is then boiled in leaden tubs with diluted sulphuric acid, which

dissolves the oxide of copper in the form of sulphate of copper; this salt is obtained in marketable crystals from the solution. The residue, which contains the silver and gold, is washed, mixed with half its weight of litharge, and made up into balls, which are dried and smelted together with more litharge and lead-slugs, when the lead obtained contains all the silver and gold, which are recovered from it by cupellation.

Very poor ores have sometimes been treated by melting them, either in cupola or reverberatory furnaces, with iron pyrites, which takes up the silver. By smelting this pyrites together with galena, the silver is obtained with the lead, which may be separated from it as usual (p. 320).

A considerable quantity of silver is now extracted by Claudet's process from Spanish pyrites, after it has been calcined for its sulphur, in the manufacture of sulphuric acid. This process is described under COPPER.

Applications of Silver.—Pure silver is far too soft to resist the wear to which it is subjected in common use. It is therefore hardened by alloying it with copper, a considerable quantity of which may be added without material alteration of colour. The hardest alloy is that which contains 4 parts of silver and 1 part of copper. The standard silver used for coin and for silver articles in England contains, in 100 parts, $92\frac{1}{2}$ parts of silver and $7\frac{1}{2}$ parts of copper; the French silver coinage contains 90 parts of silver and 10 parts of copper. When the copper exceeds this amount, it is oxidised when the alloy is exposed to the air, whence the tarnished appearance of the silver coinage of Prussia, which contains one-fourth of copper.

In English commerce, the purity or fineness of silver is generally expressed as so many pennyweights (dwts.) better or worse than the standard silver, of which the troy pound contains 11 ozs. 2 dwts. of pure silver and 18 dwts. of copper (commonly called *alloy*). Thus the French coin, which contains 24 dwts. of copper in the troy pound, would be

described as *worse* 6 dwts., because it contains that quantity less silver than the English coin. Mexican dollars contain $23\frac{1}{2}$ dwts. of copper in the troy pound, being *worse* $5\frac{1}{2}$ dwts. Indian rupees sometimes contain only 12 dwts. of copper in the troy pound; hence they are *better* 6 dwts., that is, they contain 6 dwts. more silver than the English standard.

The specific gravity of English silver coin is 10.3, and since all the alloys used to make counterfeits have tin for their chief constituent, they have usually a lower specific gravity, so that the best method by which to test whether a florin, for example, is good, without injuring it, is to ascertain its specific gravity, by weighing it first in the ordinary way, and afterwards when suspended in water, and dividing its weight in air by the loss of weight in water, when the quotient, for a genuine coin, would be 10.3. Of course, if the coin be new, it will be sufficient to ascertain that it weighs just as much as a good florin, and is of exactly the same diameter and thickness, which are readily measured by cutting a slit in a piece of cardboard through which a new florin will exactly pass.

Standard silver is whitened by being heated until the oxygen of the air has converted a little of the copper at the surface into oxide of copper, which is dissolved off by immersing the metal in weak vitriol (diluted sulphuric acid) or in ammonia, or by boiling it in a solution of cream of tartar and common salt. The film of nearly pure silver which then remains at the surface exhibits a want of lustre and is called *dead* or *frosted silver*. It is brightened by burnishing.

Oxidised silver, as it is erroneously called, is made by immersing articles of silver in a solution obtained by boiling sulphur with potash, when the metal becomes coated with a thin film of sulphuret of silver.

The tarnish which is produced upon the surface of silver when exposed to air is also due to the formation of a coating of sulphuret of silver by the action of sulphuretted hydrogen;

the sulphuret of silver is itself black, but a thin film of it upon the surface of the metal often exhibits the rainbow colours caused by the decomposition of the light reflected through it. Tarnished silver is most readily cleaned with a solution of cyanide of potassium, but this salt is so fatally poisonous that its general use should be discouraged. Ammonia (hartshorn) will also remove the film of sulphuret of silver if assisted by friction.

Plated articles are made of an alloy of copper and brass coated with silver. The brass having been melted with the requisite proportion of copper in a black-lead crucible, is cast into bars 3 inches broad, $1\frac{1}{2}$ inch thick and 18 or 20 inches long. The two faces of the bar are carefully smoothed with a file, and a plate of silver $\frac{1}{8}$ th inch thick, and somewhat shorter than the bar, is laid upon each face and tied with iron wire. A little saturated solution of borax is allowed to run in round the edges, in order that this salt may melt and dissolve the oxide off the surface of the brass when heated in the furnace. The bar is now laid upon a coke fire and heated until the surfaces of brass and silver have contracted a firm adhesion, when the compound bar is ready for the rolling mill. After rolling, it is cleaned with diluted sulphuric acid. Sometimes the clean copper surface is washed over with solution of nitrate of silver before applying the plate of silver. A thin film of silver would then be chemically deposited upon the surface of the copper and would both prevent oxidation and favour the adhesion of the silver plate.

The *plated wire* used for making toast-racks, &c., is made of copper coated with silver. The strip of silver is bent round into the form of a hollow cylinder, its edges somewhat overlapping; a red-hot cylinder of copper is thrust into this, and the edges of the silver are joined together by rubbing with a steel burnisher. The copper core is then withdrawn, the tube of silver thoroughly cleaned inside, and slipped upon a bright copper rod so as to fit closely, leaving the ends of the

copper somewhat projecting. Grooves are made in these ends, into which the silver is forced down, so as to exclude the air from the copper surface inside. The cylinder is made red hot and well rubbed with a steel burnisher, until the silver thoroughly adheres to the copper, which is then drawn into wire.

Electro-plating is now very generally employed for coating articles of baser metal with a film of silver. This art consists in decomposing a solution containing silver, with the aid of a galvanic battery, in such a manner that the metal may be deposited upon the surface of the article to be plated ; German silver (p. 298) is generally employed as the material

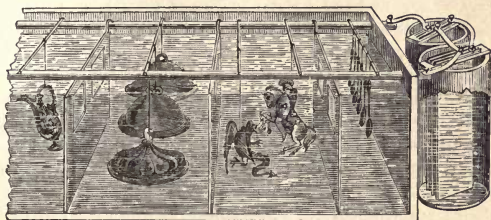


FIG. III.—Process of Electro-plating.

for the latter, the articles being then said to be *electro-plated on white metal*. They must be thoroughly cleaned by boiling them with soda, washing with water, and dipping into very weak *aquafortis* (dilute nitric acid) to take off the film of oxide ; they are afterwards washed, scoured with sand, again dipped in the weak acid, and finally rinsed in water.

The articles to be electro-plated are suspended by stout copper wires (Fig. III) in a vessel of wood or earthenware containing a solution of cyanide of silver in cyanide of potassium, of which every gallon contains an ounce of silver. The suspending wires are connected by stout copper wires with the last zinc plate of a galvanic battery consisting of

alternate plates of zinc and copper immersed in diluted sulphuric acid; the last copper plate of this battery is connected with a series of silver plates suspended in the silvering liquid opposite to the articles to be coated. The galvanic influence (or *current*) transmitted from the battery causes the decomposition of the cyanide of silver in the solution, the silver being deposited upon the articles to be plated, and the cyanogen, which was combined with the silver in the liquid, uniting with the silver upon the plates of that metal and forming a fresh quantity of the cyanide of silver equal to that which has been decomposed; this dissolves in the liquid and always maintains it of the same strength. The articles are weighed before and after plating, in order to ascertain the amount of silver which has been deposited upon them; it usually amounts to about an ounce and a half upon a square foot of surface.

In order to secure the perfect adhesion of the film of silver, the objects to be plated are sometimes dipped into a solution of nitrate of mercury until they are covered with a thin coating of that metal, before they are immersed in the silvering-bath. They are then *struck* by placing them in the silvering liquid and connecting them with the zinc of a strong battery for a short time, after which they are brushed with fine sand to show that the coating is perfect, and the process of silvering is then proceeded with.

To preserve the silvering solution of uniform strength, the articles to be plated are sometimes kept in motion by attaching the connecting rods to a frame furnished with wheels which travel along a rail on the edge of a vat, and are moved by clock-work or steam-power.

The deposit of silver is without lustre and requires bur-nishing. It is dried by immersing the article in boiling distilled water, and allowing it to dry by its own heat when removed. When a lustrous deposit is required, one gallon of the silvering liquid is mixed with six ounces of a liquid called *bisulphide of carbon* and set aside for twenty-four

hours. Two ounces of this solution are added to twenty gallons of the silvering liquid, and left for twelve hours before use. The action of the bisulphide of carbon in causing a lustrous deposit has not yet received a satisfactory explanation.

Silvering for merely Ornamental Purposes.—Where a very thin film of silver only is required, as for articles not subjected to much wear, advantage is taken of the great malleability of this metal, in which it is surpassed only by gold, to beat it out into exceedingly thin leaves, which are applied to the surface to be silvered. The silver leaf is manufactured in the same manner as gold leaf, to which the reader may refer. It is applied to non-metallic objects with some adhesive liquid, such as gum or size. In covering metallic objects with silver leaf, they are heated to remove grease, and plunged into weak aquafortis to dissolve off the oxide. The surface is next scoured with wet pumice stone, warmed, and again dipped in weak aquafortis, to roughen it, so that the silver leaf may more readily cling to it. If necessary, the surface is further roughened by hatching with a graving tool. The metal is then carefully heated till a thin film of oxide causes it to assume a bluish tint, the leaves of silver applied in successive layers, and well fixed by a burnisher of steel, the object being heated again before every application of the silver.

The process of *dry silvering* upon copper and brass, which is now seldom followed, consisted in applying to the clean surface an amalgam of silver from which the mercury is afterwards expelled by heat. A pasty amalgam is made by dissolving silver in about six times its weight of mercury. The amalgam is applied with a brush made of brass wire dipped into a solution of nitrate of mercury, which, being decomposed by the copper and zinc of the brass, deposits a coating of mercury upon the brush to which the silver-amalgam then readily adheres. The article is then moderately heated to expel the mercury in vapour, when a dead

film of silver is left upon the surface, which is afterwards burnished.

For silvering flat surfaces, such as the scales of barometers, the chloride of silver is employed. To prepare this, a piece of standard silver is dissolved in a glass or earthen vessel, with the aid of heat, in *aqua fortis* (diluted nitric acid). If any dark powder remains undissolved, it consists of finely divided gold, which is often found in old silver. The solution, which contains nitrate of silver and nitrate of copper, is mixed with common salt dissolved in water. The chloride of sodium (common salt) decomposes the nitrate of silver, forming nitrate of sodium, and chloride of silver, which separates as a white curdy *precipitate*. The liquid is well stirred, the chloride of silver allowed to settle down, the liquid poured off and replaced by fresh water; after this has been repeated several times, the washed chloride of silver is dried in an oven, and finely powdered. One part of the powder is mixed with three parts of pearlash, one part of chalk, and one-and-a-half of salt. The surface of the copper or brass to be silvered is rubbed with a wet cork or leather dipped in this mixture, when the metal decomposes the chloride of silver, forming chloride of copper, and, in the case of brass, chloride of zinc, and metallic silver is deposited.

A mixture of the chloride of silver with ten parts of cream of tartar is said to answer the same purpose.

Silvering on glass is effected by precipitating silver from a solution, in contact with the glass, by certain chemical agents, and being a purely chemical process, will not be considered here. Looking-glasses are silvered with an amalgam of tin (see *Mercury*).

GOLD.

There is no positive evidence that gold exists in Nature in any other than the metallic state, though it is believed by some to exist as a sulphuret in some varieties of pyrites. There are few regions in which small quantities of this metal cannot be discovered, though in the great majority of cases its quantity is too small to pay for the labour of separating it from the other matters with which it is associated.

In England, small quantities of gold are found in the Cornish alluvial deposits which furnish the stream tin ore. In Wales, it has been found near Dolgelly. Ireland has furnished gold from Wicklow, where it is found scantily distributed through sands in the form of gold-dust, and very rarely in small rounded fragments or *nuggets*. In Scotland, the precious metal has been traced in Perthshire, and, very recently, considerable quantities of it have been extracted in Sutherlandshire.

On the continent of Europe, the gold mines of Hungary and Transylvania are the most important. At Königsberg, the metallic gold is disseminated through sulphuret of silver.

In Sweden, gold is found associated with pyrites at Edelfors in Smoland.

The sands of the Rhine contain minute quantities of gold for which they are sometimes washed when work is scarce, although about eight million parts of sand must be washed for one part of gold.

In Spain, the province of Asturias formerly furnished a considerable quantity of gold, but the workings are now neglected.

Italy is by no means destitute of gold. Veins of pyrites containing gold are found in a granitic rock at the foot of Monte Rosa. The sands of some of the rivers on the southern slopes of the Alps also furnish gold.

Siberia yields gold distributed through *hornstone*, a variety of quartz. The sands of Siberian rivers are not considered

to be worth washing if they contain less than one part of gold in a million.

The Ural mountains contain some rich gold districts, the metal being found in pyrites, in clay, and in the sands of rivers.

Japan, Ceylon, Borneo and Thibet also contribute to the supply of gold.

Africa seems to have been the oldest and richest of the sources of gold. Sofala, on the coast of Caffraria, has sands abounding in gold-dust, and is reputed to have been the Ophir of the ancients. To the south of the great desert of Sahara, the negroes dig out earth rich in gold-dust to a considerable depth.

In modern times, down to about the year 1850, Brazil, Chili, Peru and Mexico purveyed most of the gold employed throughout the world. Minas Gerães in Brazil was a celebrated auriferous district.

But the discoveries of gold in California and Australia, which are yet fresh in the memory of the present generation, have immensely increased the supplies of the metal. In California, the gold is chiefly *alluvial gold*, being found in the alluvial deposits formed by the Sacramento and other rivers. In Australia, *gold-quartz* is more common, the metal being disseminated in thin plates, and in branch-like fragments, through lumps of quartz-rock. It is also abundant, in the form of gold-dust and nuggets, in the alluvial formation produced by the crumbling down of the rocks containing gold, under the influence of torrents which have carried the gold, together with clay and other matters, into deep gullies, at the bases of the rocks, where the *alluvium* has been deposited upon a bed of pipe-clay, being richest in gold at the lower part of the deposit, on account of the great weight of the metal.

Native gold always contains silver, and generally copper, but in very variable proportions, though the gold from the same district has commonly the same composition. Australian gold is remarkably pure.

The simplest method by which gold is extracted is that of washing the alluvial deposits, the sands of rivers, &c. There are various modes of effecting this, according to the resources of the gold-washers, but in all cases the separation of the gold from the earthy matters depends upon the high specific gravity of the gold (19.3), the lighter earthy matters being carried off by the water.

In Africa, the deposits containing gold are washed by the negroes in the shells of gourds, being well stirred up with water, which is poured off with the earthy matters suspended in it, leaving the gold dust in minute flattened grains at the bottom. The gold is kept in tubes made from the quills of the ostrich or vulture.

In South America, the washing is conducted in shallow iron or zinc pans (Fig. 112).



FIG. 112.—Gold-washing Pan.

Sometimes the deposits are thrown upon the top of a sloping plank with shallow grooves cut across it, when the grains of gold settle down into the grooves, and the earthy matters are carried on by the stream.

Long shallow troughs of wood are employed by some gold-washers, lined at the bottom with coarse baize, or with tanned skins with the hair upwards. The grains of gold become entangled in these, and when the earthy matters have been washed away, the linings are well beaten over a tank of water to remove the gold.

At the Californian and Australian gold-diggings, a *cradle* (Fig. 113) has been extensively employed. This is a wooden trough about six feet long, resting upon rockers. At the head of it is a grating upon which the alluvial deposit to be washed is thrown. This end of the cradle is about four inches higher than the other, so that a stream of water entering it flows through and escapes at the lower end, left open for this purpose, carrying the earthy matters with it, leaving

the particles of gold, with a small quantity of earthy matter, in the trough. These are swept out into a pan, dried in the sun, and freed from the lighter matters by blowing upon them.

In the gold washings of the Ural Mountains, the sands are thrown into boxes the bottoms of which are made of

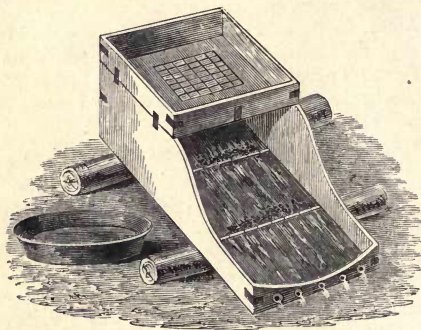


FIG. 113.—Cradle for Gold-washing.

perforated iron plates. These boxes are placed under a fall of water, in which the sands are stirred up with a shovel. The fine sand and gold dust are washed through on to sloping boards covered with baize, a workman being engaged in sweeping the deposit up the inclined plane with a heather broom. After a second washing on a smaller inclined table, the particles of magnetic iron ore are extracted by a magnet, and the gold dust is melted in a plumbago crucible, when the earthy matters remain upon the surface of the melted gold.

At some of the Russian works, the sifting of the sands is effected in cylinders of perforated sheet iron, which are placed in a sloping position above the washing tables, and made to rotate upon an axis. A stream of water being let in at the upper end, carries the sand and gold dust through

the perforations on to the sloping tables, while the large pebbles pass out at the lower end of the cylinder.

The boxes in which the more tenacious alluvial deposits are mixed with water are sometimes provided with agitators worked by horse or steam power, and having knives attached for breaking up and mixing the deposits.

When the gold is disseminated through quartz or some similar rock, this must be crushed in order to extract the gold, an operation attended with great expense, on account of the hardness of the rock. Where it is possible, the rock is rendered more brittle by being heated to redness and quenched with water. The crushing is effected either by passing the gold-quartz between chilled cast-iron rollers, or by means of stampers similar to those employed in the Cornish tin-works.

From the stamped ores, as well as from the auriferous sands which have been concentrated by washing, the gold is sometimes extracted by a process of *amalgamation* similar to that employed in the case of silver.

As practised by the Mexican gold-washers, the process of amalgamation consists in shaking the damp gold-dust, still mixed with foreign matters, with metallic mercury which dissolves the gold. The impurities having been washed away, the amalgam is squeezed in a cloth, when about half the mercury flows out, and the solid amalgam remaining in the cloth is placed in a small iron dish, covered up with green leaves and set over a charcoal fire; a good deal of the mercury vapour is condensed in the leaves, which are renewed from time to time as they get dry.

In the Tyrol, particles of gold exist disseminated through iron pyrites, from which they are extracted by amalgamation. The amalgamating mill (Fig. 114) is a large cast-iron dish (*e*) firmly fixed upon a wooden table. In this dish there is a heavy cone of hard wood (*m*) of the same shape as the dish, and just large enough to leave an interval of half-an-inch between them; several projecting iron ribs are fixed to the under side of this cone, which nearly touch the bottom of

the pan, and the upper surface of the cone is hollowed out so as to form a shallow funnel. The wooden cone is connected with an axle (*a*), so that it may be made to revolve at the rate of about twenty turns a minute. About 50 lbs. of mercury having been poured into the iron pan, the auriferous pyrites, previously stamped or ground to a fine powder, is brought into the mill by a stream of water from the spout (*g*), when it is thoroughly stirred with the mercury by the projecting ridges at the bottom of the wooden cone. In order that no gold may escape being dissolved by the mercury, the pyrites which has been treated in one mill

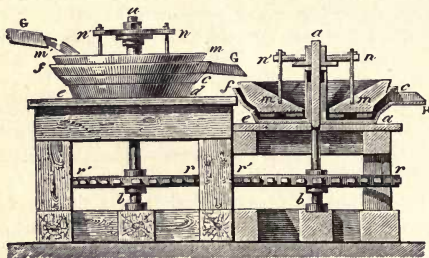


FIG. 114.—Mill for amalgamating Pyrites containing Gold. *r r'*, Toothed wheels for transmitting motion to the axle *a b*.

flows out through a spout (*g*) into the next, and so on through an entire series. After about a month, the mercury is drawn off and squeezed through wash-leather, which allows the liquid portion to pass through, and retains a soft solid amalgam containing about one-third of its weight of gold, from which the mercury is separated by distillation in the apparatus represented in Fig. 115.

For the extraction of gold from gold-quartz, lead has been employed with great advantage, since this metal, when melted, will dissolve gold just as mercury will at the ordinary temperature. The crushed quartz is fluxed by an addition of lime and clay (see *Iron*), with which is added either

metallic lead or galena (sulphuret of lead), or even rich lead slags, with some coal or charcoal to reduce the lead to the metallic state.

The lead containing gold (and silver) is then subjected to the process of cupellation (p. 336).

In Hungary, gold is extracted from iron pyrites associated with quartz, by taking advantage of the property of dissolv-

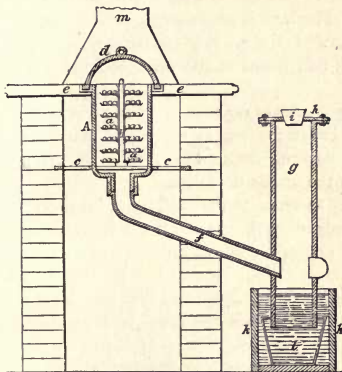


FIG. 115.—Apparatus for distilling the Amalgam of Gold. *a*, Dishes for receiving the amalgam, attached to the pillar *b*. *A*, Iron cylinder heated by a fire on the grate *c*. *d*, Iron dome. *e*, Iron cover of the furnace. *f*, Tube for vapour of mercury. *g*, Cylinder in which the mercury condenses, closed by an iron plate *k* and a wooden plug *i*. *l*, Water-tank. *l*, Vessel for collecting the mercury. *m*, Chimney.

ing gold possessed by sulphuret of iron. The pyrites is roasted in heaps with brushwood, to convert a part of the bisulphuret of iron into oxide of iron, and a part into sulphuret of iron. The roasted ore is fused with an addition of lime, when a slag is formed by the combination of the silica (quartz), the oxide of iron, and the lime, whilst the sulphuret of iron fuses, dissolves the gold, and forms a matt beneath the slag.

This sulphuret of iron is roasted so as to convert it into

oxide, and fused with a fresh quantity of the auriferous pyrites and the requisite proportion of lime, when a fresh quantity of the matt will be obtained, containing the gold from the two charges of pyrites. This operation is repeated until a sufficient quantity of gold has accumulated in the matt of sulphuret of iron, which is then melted down with lead; this metal extracts the gold from the sulphuret of iron, and the latter remains in a melted state upon the surface of the lead. The lead is afterwards cupelled in order to extract the gold, and since lead containing silver is generally employed, this metal is left on the cupel alloyed with the gold.

Whenever ores of copper, lead, or silver contain gold, the latter is always present in the metal extracted from them, and is recovered from those metals by the processes described in the article on Silver.

In order to separate the gold from the silver with which it is commonly alloyed, whether it has been obtained by washing, or by any of the above processes of extraction, the alloy of silver and gold is heated with sulphuric acid, which converts the silver into sulphate of silver, capable of being dissolved by water, and leaves the gold untouched.

Parting by Sulphuric Acid.—The alloy of silver and gold, in which, of course, the former metal always predominates, is melted either in wrought-iron or plumbago crucibles, and poured into water in order to *granulate* it or divide it into a flaky condition, exposing a large surface to the action of the acid. The granulated metal is dried, weighed, and boiled with oil of vitriol (concentrated sulphuric acid). When the alloy is rich in gold, the operation is performed in platinum alembics or stills, but in the more common case, where the silver contains only a few grains of gold in the pound, cast-iron pans are employed; each pan (about two feet wide) has an iron lid, from which a bent pipe passes down into an airtight leaden tank, where the vapour of sulphuric acid which escapes during the boiling may be condensed. A large quantity of sulphurous acid gas passes off during the opera-

tion, and this is conducted, from the leaden tank, by a pipe of the same metal, into a large leaden chamber, 30 feet long by 10 feet wide, and 6 feet high, in which it is reconverted, by an appropriate chemical process, into oil of vitriol, which is used over again.

One fire is made to heat two of the iron pots, in which the granulated silver is placed, together with twice its weight of concentrated sulphuric acid, which is gently boiled until the silver is entirely converted into sulphate of silver, forming a pasty mass consisting of minute crystals. This is taken out by cast-iron ladles, and thrown into leaden cisterns, where it is stirred up with water, and boiled by passing steam into it through perforated leaden pipes connected with a boiler. The boiling water dissolves the sulphate of silver, and the finely-divided gold is left as a black powder, which, when accumulated in sufficient quantity, is well washed and dried. It still retains a small proportion, varying from $\frac{1}{30}$ th to $\frac{1}{80}$ th of its weight, of silver.

The solution of sulphate of silver is drawn off, by leaden siphons, into leaden troughs, where it is left in contact with shavings of copper. This metal enters into solution, forming sulphate of copper, and separating the silver in a finely-divided state, as a grey powder; this is allowed to settle down, the solution of sulphate of copper run off into another cistern, and the silver washed with fresh water, drained, and compressed by hydraulic pressure, in a square cast-iron box, which makes it into cakes of 60 lbs. each. These are dried, melted in plumbago crucibles, and cast into ingots. Cast-iron crucibles strengthened by shrinking hot iron hoops upon the cold crucibles are sometimes employed, but since they become impregnated with silver, the latter must be extracted by melting some lead in them when they are worn out.

The solution of sulphate of copper formed in displacing the silver by copper is evaporated in shallow leaden pans, to a proper strength, and the sulphate of copper allowed to crystallise out on cooling. The liquid remaining after the

last crystals have separated contains the excess of sulphuric acid which has been employed in the process, very little sulphate of copper being left in it, because this salt is almost insoluble in moderately strong sulphuric acid. This liquor is boiled down in a platinum still, until the water has boiled away, and the concentrated sulphuric acid is left in the still, ready to be employed for the treatment of a fresh quantity of silver.

The sulphate of copper (blue vitriol) obtained in this process is a salt for which there is a considerable demand; it is largely used for dressing grain intended for seed, to prevent smut. It is also employed in dyeing and calico-printing, and in many other branches of industry, as well as in several forms of galvanic battery.

When there is no market for the sulphate of copper, the solution of the salt is decomposed by scrap iron, as in the case of the blue water at Anglesea (p. 248), to recover the metallic copper.

In such works for the refining of gold and silver, the processes can be conducted economically only when great care is taken to avoid the loss of any particles of the precious metals. Thus all the old crucibles are ground and treated with mercury in the amalgamation mill, and after as much gold and silver as possible have been thus extracted, the residues are sold to the *sweep-washers*, who extract a little more by melting with lead. The very dust off the floors is collected and treated in a similar manner. One part of gold can be profitably extracted from 2,000 parts of alloy by this process of parting by sulphuric acid. Its introduction has affected the metallurgy of gold in the same way as Pattinson's process did that of silver, much old silver plate having been treated by it for the sake of the gold which had not been found worth extracting by the older and more expensive method of parting by nitric acid.

When the alloy contains copper as well as silver and gold, it may also be treated in the same way, the copper being removed, with the silver, as a soluble sulphate; but the pro-

cess does not succeed well with an alloy containing more than 75 parts of copper in 1,000, so that, if it be richer, it is either melted with more silver, or is cupelled with lead (p. 336), in order to reduce the copper to the right proportion.

Nor should the alloy contain more than one-fifth of its weight of gold, or the sulphuric acid will not extract the silver. When platinum stills are employed for parting by sulphuric acid, it is necessary that the alloy should be free from lead and tin, which are apt to melt upon the bottom of the still and seriously to corrode the platinum.

Parting of Gold and Silver by Nitric Acid.—Silver is easily dissolved by nitric acid and converted into nitrate of silver, but this acid, if pure, does not attack gold. If the nitric acid contains chlorine, however, it will dissolve some of the gold, so that it is always necessary to test it by adding a little solution of nitrate of silver, which will render it milky, from the separation of the insoluble chloride of silver, if any chlorine be present. An alloy containing more than one part of gold to three parts of silver is very little affected by nitric acid, so that it becomes necessary to fuse very rich alloys with so much silver that the gold shall form only one-fourth of the alloy; this is the origin of the term *inquartation* or *quartation*, used in speaking of this process.

The alloy, in a granulated state, is heated with twice its weight of moderately strong nitric acid (sp. gr. 1.32) in a still made of platinum, glass, or earthenware, connected with an apparatus for condensing the vapours of nitric acid which pass off. Whilst the silver is being dissolved, a large quantity of red gas is evolved, resulting from the action of the silver upon the nitric acid, and when this is no longer perceived, the silver is known to be dissolved. The still is then cooled, the solution of nitrate of silver drawn off, and the undissolved gold boiled with a little more nitric acid to extract any remaining silver. It is then washed with water, dried, melted, and cast into an ingot.

In order to recover the silver from the nitrate, hydro-

chloric acid is cautiously added, so as to separate the bulk of the silver as the insoluble chloride, leaving the nitric acid in the solution, which may be used again, if care be taken to leave a little nitrate of silver undecomposed in the solution, so as to ensure the absence of chlorine. The separated chloride of silver is washed with water, moistened with sulphuric acid, and some bars of zinc placed in it, when chloride of zinc is formed and dissolved, the silver being left in the finely-divided metallic state. The rest of the zinc is then taken out, the silver allowed to remain in contact with dilute sulphuric acid to dissolve any particles of zinc, then thoroughly washed with water, dried, melted, and cast into ingots.

Refining of Gold.—The gold obtained by parting with sulphuric acid is refined by mixing it with one-fourth of its weight of dried sulphate of soda, and treating it, in an iron pan, with oil of vitriol, to the amount of three parts for every five parts of sulphate of soda. Heat is applied as long as any vapours of sulphuric acid escape. This is repeated a second time, but without driving off the whole of the sulphuric acid. The mass is then boiled with sulphuric acid, when the gold alone is left, and is melted with a little saltpetre, which extracts a little platinum, before casting it into an ingot. At the Russian mint, the re-melting is effected in a small reverberatory furnace, with a cavity in which the gold collects. The explanation of this process is simply that the sulphuric acid combines with the sulphate of soda, and may then be raised to a higher temperature without vaporising than is possible with uncombined sulphuric acid. The higher temperature employed enables the sulphuric acid to attack the remainder of the silver.

Extraction of Gold from Gold-quartz in the wet way.—It has been proposed to avoid the expensive process of amalgamation, by digesting the pulverised gold-quartz with $\frac{1}{100}$ th part of black oxide of manganese and some muriatic acid, in an earthen vessel, for twelve hours, when chlorine is generated, which dissolves the gold in the form of chloride of gold, from

which the metal may be separated in a finely-divided state by adding to the liquid a solution of copperas (sulphate of iron). The dark powder of gold thus separated is washed with water, dried, and melted down. When the gold contains much silver, common salt is employed to dissolve the chloride of silver, which would otherwise protect the gold from the action of the chlorine.

Plattner's Process, which was employed with economy for the extraction of gold from the abandoned residues of roasted pyrites at Reichenstein, in Upper Silesia, containing less than 1 oz. of gold per ton, is rather a chemical than a metallurgical process, and consists in treating the fine powder, in a moist state, with chlorine gas, which converts the gold into a soluble chloride; this is washed out with water, and treated with sulphuretted hydrogen, which separates the gold as an insoluble black sulphuret, leaving the iron, &c., in the solution. The sulphuret of gold is heated to expel a part of the sulphur, dissolved in a mixture of hydrochloric and nitric acids, and separated from the solution in the pure metallic state by sulphate of iron. The precipitated gold is washed, and melted down with a little borax and saltpetre.

Gold dust and nuggets of gold never consist of the pure metal, but always contain silver, and sometimes copper and small quantities of other metals, such as antimony and bismuth. Grains of platinum and its allied metals are also very commonly found in alluvial gold. The purest native gold has been found at Giron, in New Grenada, containing only $\frac{1}{100}$ th part of silver. Some Californian gold contains as much as nine parts of silver and nearly one part of copper in a hundred parts. Californian gold also sometimes contains small grains of an extremely hard alloy of osmium and iridium, which occasion great injury to the die in coining, since they remain unchanged when the gold is cast into ingots.

At the American mint, the Californian gold, which contains about $\frac{1}{1000}$ th of its weight of the osm-iridium alloy, is melted with thrice its weight of silver, which lowers its specific gravity, and allows the osm-iridium to settle to the

bottom. The greater part of the melted metal is ladled out, leaving the rest very rich in osm-iridium at the bottom. This is repeatedly melted with silver, by which the proportion of gold is still further diminished, and ultimately the mixture of osm-iridium with silver and a little gold is boiled with sulphuric acid, which extracts the silver, leaving the osm-iridium mixed with some powdered gold, which may be removed by washing.

When gold contains platinum or palladium, it is very much lighter in colour. These metals cannot be separated by the ordinary refining processes. The presence of lead or antimony, even in very minute proportion, is found to render gold extremely brittle.

Australian gold is frequently brittle from the presence of lead or antimony. It is sometimes refined by stirring a little corrosive sublimate (chloride of mercury) into the melted gold, when the chlorine combines with the base metals, forming chlorides which are expelled in the form of vapour, together with the liberated mercury. F. B. Miller has introduced an improved process for refining such gold by forcing into the melted metal a current of chlorine gas, through the stem of a tobacco-pipe. The chlorine converts the silver present in the gold into chloride of silver, which collects, in a melted state, upon the surface of the gold, whilst any arsenic, antimony, bismuth, lead, or zinc, is also converted into chloride, and driven off in the form of vapour. The silver is afterwards easily extracted from its chloride.

Perfectly pure or *fine gold* is nearly as soft as lead, far too soft therefore to resist the wear to which it would be subjected in coinage and gold plate. The alloy used for coin, in England, consists of 11 parts of gold and 1 part of copper, which is harder and more fusible than pure gold. Formerly, the gold was alloyed with silver, or with silver and copper, and this latter alloy is still employed by goldsmiths. The guinea was composed of 11 parts of gold, $\frac{1}{2}$ part of silver, and $\frac{1}{2}$ part of copper. The specific gravity of sovereign

gold is 17·157 (that of pure gold being 19·3). The safest method of ascertaining whether a sovereign is genuine consists, as in the case of silver coin (p. 361), in showing that it has the same size and weight as a sovereign known to be genuine. A new sovereign weighs $123\frac{1}{4}$ grs., but it is a legal tender as long as it is not less than $122\frac{1}{2}$ grs. in weight. So hard is the alloy, that with proper wear, a sovereign will circulate for eighteen years without falling below the legal standard.

The gold coin of the United States and of France contains only 9 parts of gold to 1 part of copper.

The fineness or purity of gold is commonly expressed by stating how many carats of gold are present in 24 carats of the alloy. Thus pure gold would be 24 carats fine; sovereign gold, 22 carats fine.

Fractions of a carat are expressed in grains (4 grains are equal to 1 carat) and eighths of a grain; thus French gold coin would be styled of 21 carats $2\frac{3}{8}$ grs. fine, or *worse O carat* $1\frac{5}{8}$ grs., implying that it contained so much less gold than the English standard.

The fineness may of course be judged of, as in the case of sovereign gold, from the specific gravity, since the specific gravities of silver and copper (respectively, 10·5 and 8·9) are so much lower than that of gold (19·3). But this test has been found fallacious in a case where bars of platinum (sp. gr. 21·5) were coated with gold and sold as solid ingots of that metal, which is more than twice the price of platinum. Gold of 18 carats fine has the specific gravity 16·8.

The goldsmith or pawnbroker generally tests the gold by touching its surface with a stopper wetted with *aqua fortis* (nitric acid), which produces a green stain upon the metal when a very large proportion of copper is present, nitrate of copper being then produced. This is, at best, a rough test, and would of course fail altogether if the surface only of the base alloy were coated with fine gold.

The use of the *touchstone* admits, in practised hands, of a

far more exact determination of the value of the alloy, and unless it be pretty thickly coated with richer metal, deception is more easily detected. The touchstone is a piece of black basalt, or even of black slate, over which the gold to be tested is drawn so as to leave a streak of fine particles of the metal upon the surface; this streak of course remains untouched when moistened with nitric acid, but if a streak of any base alloy (of copper and zinc for example), made to imitate gold, be made upon the surface of the touchstone, the nitric acid will immediately dissolve it.

The acid employed in this test is generally mixed with a minute proportion of hydrochloric acid (98 parts by weight of nitric acid, of sp. gr. 1.34, 2 parts of hydrochloric acid, of sp. gr. 1.173, and 25 parts of water). The streak is not apparently affected by the acid if the gold is not below 18 carats fine; by making several streaks in succession, or by grinding off a part of the surface upon the touchstone, any error arising from a thin external coating of fine gold may be avoided; the feather of a pen, or a glass rod, serves for moistening the streaks with the acid.

In order to determine by the touchstone the proportion of gold which is present in the alloy, the streak is compared with that made by a series of *touchneedles* composed of alloys containing gradually diminishing quantities of gold. In experienced hands, the quantity of gold may thus be ascertained with an error of not more than one part in a hundred.



FIG. 116.—Cupel.

The exact assay of the alloys of gold is an imitation, on the small scale, of

the metallurgic processes of inquartation, cupellation and parting. A weighed quantity of the alloy, say 6 grs., is wrapped in a piece of thin paper together with three times its weight of pure silver, and added to twelve times its weight of pure lead already melted in a bone-ash cupel

(Fig. 116), heated in a muffle (Fig. 117) or arched clay oven, with slits for admitting air, which is placed in a brisk fire (Fig. 118). The lead and copper are both converted into oxides, the oxide of copper dissolving in the oxide of lead, and both being absorbed by the bone-ash. After the *brightening* (p. 339) the alloy of silver and gold which is left on the cupel is hammered flat, annealed by heating to redness, rolled out thin, coiled up into the form of a tube, and boiled, first with weak nitric acid (sp. gr. 1.18), and then with a stronger acid (sp. gr. 1.28) in order to extract the silver; the gold is left in the form of a little tube (*cornette*) having much the appearance of red earthenware. It is well washed with water, carefully transferred to a small crucible without breaking it, dried, and heated to redness in the muffle, when it shrinks, and assumes the ordinary appearance of gold. Its weight is ascertained by a very accurate balance, and multiplied by four (six grains having been assayed) to express the fineness of the gold. To avoid errors, the exact assayer commonly passes a *proof* or weighed quantity of pure gold through the same process, at the same time, with the addition of a proportion of copper about equal to that in the alloy, and corrects his assays by deducting from the weight of the gold finally obtained the increase which the pure gold was found to have experienced in consequence of its having retained traces of lead, silver and copper.

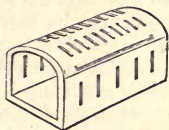
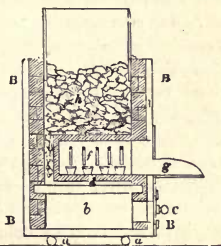


FIG. 117.—Muffle.

FIG. 118.—Assay by Cupellation. *a*, Iron castors on which the furnace moves. *b*, Ash-pit. *c*, Damper. *d*, Grate. *f*, Muffle containing the cupels. *g*, Mouth-plate. *h*, Fuel.

The alloy of copper and gold is much redder than pure gold ; the addition of silver whitens it, and its surface may be brought to any shade of gold colour by heating it until a portion of the copper is oxidised, and dissolving out this oxide with an acid. Goldsmiths commonly *colour* their gold by boiling it for twenty minutes with a mixture of 1 part of common salt, 2 parts of saltpetre, 1 part of alum and 4 parts of water, the mutual action of which would result in the production of a little hydrochloric acid (from the chloride of sodium, common salt), and a little nitric acid (from the saltpetre, nitrate of potash); this *pickle* dissolves not only the copper, but some of the gold, which is recovered by precipitating it with solution of copperas (p. 379).

The great value of gold, and its perfect freedom from alteration by exposure to the atmosphere, have led to many devices for making the smallest quantity of the precious metal cover the greatest extent of surface, by taking advantage of its extreme malleability and ductility, in which it far surpasses all other metals.

Gold-beating.—Pure gold may be extended by hammering so as to present a surface 650,000 times as large as it originally possessed, but for this purpose it must be perfectly pure or fine gold, a very small quantity of any other metal materially injuring its malleability. Gold alloyed with copper or silver is, however, often beaten into moderately thin leaves, when a colour different from that of fine gold is required, but it is of course much more liable to tarnish. Dentists' leaf gold for stopping teeth is generally beaten from fine gold, because it is more easily pressed into a compact form.

The gold is melted in a crucible with a little borax, which prevents it from sticking to the crucible, and poured into a small cast-iron mould, warmed and slightly oiled, in which it forms an ingot of $\frac{3}{4}$ inch in width, and 2 ozs. in weight. This ingot is annealed by burying it in hot ashes, and hammered upon a steel anvil until its thickness is reduced to

$\frac{1}{8}$ th inch, the metal being once or twice re-heated or annealed during the process. The gold is then passed between perfectly cylindrical steel rollers, which are turned at exactly the same rate in opposite directions, until it is reduced to a riband so thin that a square inch of it weighs $6\frac{1}{2}$ grains. It is of course necessary to anneal the gold several times during the rollings.

The riband is now exactly divided, with the help of a pair of compasses, and cut up into pieces of one inch square. One hundred and fifty of these pieces are piled up alternately with pieces of tough paper or vellum (prepared calf-skin) four inches square, rubbed over with a little fine plaster of Paris, to prevent the gold from sticking. Twenty vellums are placed above, and twenty below the pile, which is then firmly secured by passing two strong belts of parchment across it.

The beating is performed with a hammer weighing about 16 lbs., having a circular face four inches in diameter and somewhat convex. The pile is placed upon a heavy block of marble, nine inches square, sunk into a very strong wooden bench. The workman uses the hammer with one hand, directing the blows well in the middle of the pile, and turning it occasionally. After a time, the packet is opened, the middle leaves are shifted to the outside, and the beating is continued until the leaves have been extended to nearly the same size as the vellums. They are then taken out of the pile, and each leaf is cut with a sharp knife into four equal squares, which will measure about an inch each way. These are made into packets, as before, with *gold-beaters' skin*, a membrane which is separated from the outer surface of the intestines of the ox, and is prepared for use by hammering it between folds of paper in order to extract the grease, and steeping it in an infusion of nutmeg and cinnamon, in order to preserve it; it is then dried and rubbed over with plaster of Paris. The packets made up of gold-

beaters' skin, with alternate layers of gold, are beaten as before, but with a ten-pound hammer, for about two hours, the packet being skilfully rolled and bent now and then, in order to loosen the leaves. When these are again extended to about 4 inches square, they are spread out upon a leathern cushion, and cut into four equal squares by applying a cross of cane cut to sharp edges, and fixed upon a board. These squares are again made up into packets with gold-beaters' skin, and hammered out a third time, with a seven-pound hammer, to about $3\frac{1}{2}$ inches square. They are then dexterously lifted off the skin, with a delicate pair of wooden pincers, spread out upon a leathern cushion by blowing them flat down, and cut down to one size by a square of sharp-edged cane fixed to a board, and pressed upon the leaf of gold extended on the cushion. The gold leaves are then packed between the leaves of books made of printed paper, rubbed over with red chalk to prevent adhesion ; there are usually twenty-five leaves in each book, their average thickness being $\frac{1}{282000}$ th of an inch. Mechanical power has been lately substituted for manual labour in gold-beating.

When one of these leaves is held up to the light, it exhibits a beautiful green colour, and if it be rendered still thinner, either by beating, or by floating it upon a very weak solution of cyanide of potassium, which slowly dissolves it, it transmits, when taken upon a glass plate and held up to the light, a blue, violet, or red light,* in proportion as its thickness diminishes. Even when it is so transparent that one may read through it, the yellow colour and lustre of the gold are still visible by reflected light. These varying colours of finely-divided gold are turned to account in the colouring of glass and in painting on porcelain.

Gold Thread is made by covering a cylinder of silver with gold leaf, and drawing it through a wire-drawing plate until it is reduced to the thinness of a hair. In this manner, one grain of gold is made to cover 364 feet of wire. For making gold lace, this wire is flattened by passing it between

rollers, and is twisted by machinery round a thread of yellow silk, which is then made up into lace or braid.

Gilding.—The most obvious method of gilding consists in applying gold leaf to the object required to be covered, an art requiring great delicacy and skill, the description of which does not fall within the scope of this work.

Wash-Gilding is effected with an amalgam of gold, prepared by dissolving one part of fine gold in eight parts of mercury. The gold is laminated, placed in a crucible, heated to faint redness, and thrown into the requisite quantity of mercury, also moderately heated in a crucible, whilst the metals are stirred with an iron rod hooked at the end. When all the gold is dissolved, the amalgam is poured into water, and squeezed in wash-leather to separate the excess of mercury. A pasty amalgam is thus obtained, containing about one-third of its weight of gold.

The metals generally employed for wash-gilding are brass, and copper alloyed with one-seventh of brass, nickel being sometimes added. The article to be gilded, after being heated to redness in a charcoal or peat fire and cooled slowly, is dipped in very dilute sulphuric acid, rubbed with a hard brush, washed and dried. It is then dipped in pretty strong nitric acid (sp. gr. 1.33), well washed, and dried by rubbing with bran. By this treatment, all the oxide has been removed from the surface, which has been also sufficiently corroded or roughened by the acid to favour the adhesion of the amalgam of gold. This is applied to the surface with a brush made of fine brass wire, which is dipped into a solution of nitrate of mercury made by dissolving 100 parts by weight of mercury in 110 parts of nitric acid (sp. gr. 1.33) and diluting the solution with 25 times its weight of water. The brush being wetted in this solution, which coats it with a thin film of mercury (p. 365), is rubbed upon a lump of the amalgam, which is then brushed over the article to be gilded; this is next held over glowing charcoal, and well turned about, until it is hot enough to make a drop

of water hiss, when the whole of the mercury is expelled in the form of vapour, leaving a film of dead gold upon the surface, which is then rubbed with a wet burnisher of hæmatite. Several curious processes, very difficult of explanation, are employed by persons experienced in the art of gilding, for giving the required shades of colour and degrees of lustre to the film of gold left by the mercury. One of these, intended to produce red gold, consists in coating it with *gilders' wax*, containing wax, red ochre, verdigris (acetate of copper) and alum. The object, having been coated with this composition, is strongly heated in the flame of a wood fire, quenched in water, and scrubbed with vinegar. This process probably reddens the gold by alloying it with a little copper from the verdigris.

Buttons, trinkets, &c., made of brass or copper, are coated with gold by immersing them in a boiling alkaline solution containing that metal. 2,400 grains of fine gold are dissolved in a mixture of 21 (avoirdupois) ozs. of nitric acid (sp. gr. 1.45), 17 ozs. of hydrochloric (muriatic) acid (sp. gr. 1.15), and 14 ozs. of distilled water. This solution of (chloride of) gold is mixed with 4 gallons of distilled water, and 20 lbs. of bicarbonate of potash, and boiled for two hours. The articles having been thoroughly cleaned from oxide, are suspended from a hoop of brass or copper-wire, and moved about in the boiling liquid until they are judged to have acquired a sufficient coating of gold, which is the case after less than a minute with small articles. The gold is deposited, in this process, in consequence of the copper taking its place in chemical combination in the liquid; the film is bright and not dead as in the amalgamation process, but it may be deadened, if required, by dipping the articles in solution of nitrate of mercury (p. 387) before exposure to the gilding solution, when they will become coated with mercury, which will form an amalgam with the gold deposited from the solution, and this will be left as a dead film on expelling the mercury by heat.

Electro-gilding is effected, like the corresponding process of silvering (p. 363), by immersing the thoroughly-cleansed articles to be gilded in an appropriate solution of gold, and connecting them by metallic wires with the zinc end of a galvanic battery of which the copper end is in metallic connection with a plate of gold, which is gradually dissolved, replacing the gold which has been deposited. The gilding bath is heated by steam to about 200° F., which facilitates the deposition of the metal. A solution for electro-gilding may be prepared by dissolving 1,550 grains of gold in a mixture of 14 (avoirdupois) ozs. of nitric acid (sp. gr. 1.45), 11 ozs. of hydrochloric acid (sp. gr. 1.15), and 10 ozs. of water. To this liquid, containing the chloride of gold, a solution of cyanide of potassium is added carefully, as long as any cyanide of gold is separated. This precipitate is allowed to settle, the clear liquid drawn off, and the precipitate redissolved in some more solution of cyanide of potassium. Enough water is then added to bring the solution up to five gallons.

In gilding polished iron and steel, they are heated until they assume a blue tint, and successive coatings of gold leaf are applied with a burnisher, a gentle heat being employed after every coating except the last, which is burnished cold.

MERCURY.

This metal, also called *quicksilver*, is of very rare occurrence in Great Britain, the whole of the mercury employed in the arts being imported from Austria, Spain, California, and China, the largest quantity of the metal being now furnished by California. A considerable quantity is found in the metallic state, either collected in cavities, or disseminated in

globules throughout its ore *cinnabar*,* which has the same composition as *vermilion*,† being a compound of mercury with sulphur, containing, when pure, 86 parts of mercury in the hundred. Native mercury sometimes contains silver. Cinnabar is usually met with in moderately hard dark brown masses which are very heavy (sp. gr. 8·2), and exhibit a red colour when scraped with a knife. Some specimens have a red colour without scraping, and all yield a powder of a more or less bright red colour.

Native *calomel*, a combination of mercury with chlorine, is sometimes found in very small quantity associated with cinnabar.

A rich deposit of cinnabar is reported to have been lately discovered in Borneo.

The extraction of mercury from its ore is effected, like that of zinc, by distillation, but since this metal is converted into vapour at a much lower temperature than zinc (mercury boils at 662° F.), it requires much more elaborate arrangements for the condensation of its vapour into the liquid form, and in many works a considerable quantity of the metal is wasted, on account of the imperfect character of the condensers, the escaping vapour proving most injurious to the health of the persons employed. Indeed the metallurgy of mercury, in all its branches, is lamentably injurious to health, the miners as well as the smelters being liable to salivation, and seldom living to an old age.

The mines at Almaden, a town of La Mancha, in Spain, have been the most productive, the cinnabar being found in large veins surrounded by sandstone and clay slate. The mercury is extracted by simply roasting the ore, when the oxygen of the air converts the sulphur into sulphurous acid gas, and the mercury passes off in vapour.

* Derived immediately from the Latin *cinnabaris*, applied not only to this ore, but apparently also to red-lead, and to the red gum-resin known as *dragon's blood*, from the Indian name of which, *cinoper*, the word appears to have originated.

† Probably from *vermeil*, French for red coral.

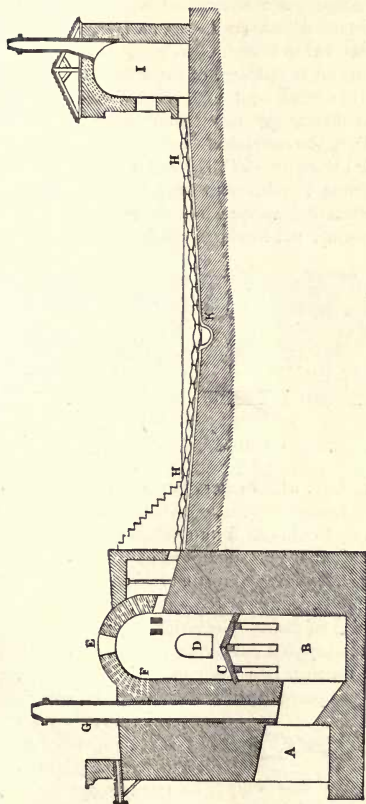


FIG. 110.—Aludel Furnace for the extraction of Mercury. A, Opening for charging the fuel. B, Fire-space. C, Arch for sustaining the ore. D E, Openings for charging the ore. F, Chamber for receiving the ore. G, Chimney. H H, Rows of aludels. I, Condensing chamber. K, Gutter for mercury.

The roasting-furnace consists of a fire-place in which a wood fire is maintained, the flame of which kindles the ore stacked upon an arch of fire-brick (c, Fig. 119), provided with openings for the passage of the flame. Upon this arch some large blocks of sandstone, very poor in cinnabar, are piled; above these, small fragments of richer ores, over which again are placed the cakes made up by kneading with clay the finely-divided mercury which has condensed from the fumes without running together into liquid metal, as well as any cinnabar which has escaped in vapour unchanged during a previous roasting, and has been condensed again as a black

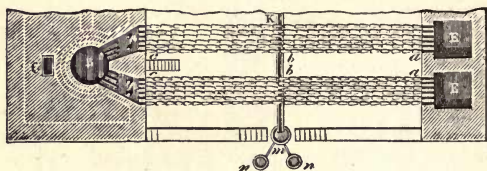


FIG. 120.—Plan of Aludel Furnace for extraction of Mercury.

powder. The air, which enters through the openings in the brick arch, furnishes oxygen to the sulphur of the sulphuret of mercury, and converts it into sulphurous acid gas, while the mercury does not combine with oxygen, but separates in the metallic state, being converted into vapour, partly by the heat of the fire beneath, partly by that produced in the combustion of the sulphur. The sulphurous acid gas and the vapour of mercury pass out through flues (ϕ , Fig. 120) in the side of the furnace, into the condensing apparatus, which consists of 300 pear-shaped stoneware pipes, called *aludels*



FIG. 121.—Aludels.

(Fig. 121), open at both ends, and fitted into each other, the joints being cemented together with clay.

These are arranged so as to form twelve separate flues each

composed of 25 aludels. The first 12 of each set, nearest to the furnace, incline downwards, towards a central gutter (κ) whilst the last twelve ascend a corresponding inclined plane the centre aludel of the series being perforated in the lower side, so that the condensed mercury may run out into a gutter which conducts it into receiving basins (m, n , Fig. 120) underneath the furnace. The vapour of mercury which escapes condensation in the aludels passes, together with the sulphurous acid gas, into a chamber (1, Fig. 119) in which a further considerable quantity of the mercury is condensed to the liquid state. The sulphurous acid gas eventually escapes through the chimney.

The imperfect character of this condensing apparatus is evident; the loss of mercury vapour through the leakage of the numerous joints, and through the openings in the lowest aludels, must be very great, so that it is not surprising that only 10 parts of mercury should be extracted from a hundred parts of rich ore. Each roasting lasts about twelve hours, and the furnace requires three or four days to cool before receiving a fresh charge, affording a striking contrast to the system of nearly continuous working adopted in the metallurgic processes of this country.

The mercury mines at Idria in Austria, which are now probably more important than those at Almaden, were formerly worked by state prisoners and criminals, on account of the unhealthy nature of the occupation. In these mines the ore is found both in limestone and in a bituminous slate, and contains a considerable proportion of bituminous matter. Previously to the year 1794, the aludel furnace just described was also employed at Idria, but it was then superseded by a very large brick structure containing a series of condensing chambers. The roasting furnace (Fig. 122) consists of a grate upon which wood is burnt, surmounted by three perforated arches ($n p r$) of fire-brick, for receiving the ore, the large fragments being placed upon the lowest arch, the smaller pieces on the next, whilst the uppermost supports a number

of shallow earthen dishes (*s*) containing the dust of the ore and the mercurial soot from former operations. Air is admitted to the heated ore through small passages in the walls,

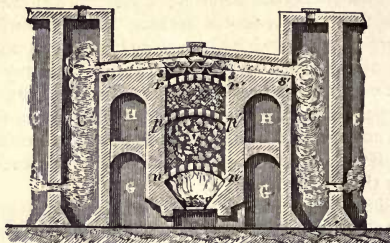


FIG. 122.—Extraction of Mercury at Idria. *G H*, Passages from which air enters through flues into the space occupied by the ore. *C*, Condensing chambers.

and the sulphurous acid and vapour of mercury are drawn by the two chimneys, through six condensing chambers (*C D*, Fig. 123) on each side of the furnace, so connected by narrow openings that the vapours are forced to pervade one

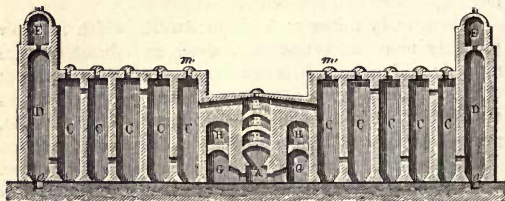


FIG. 123.—Section of Idrian Furnace and Condensing Chambers. *A*, Fire-place. *B*, Arches upon which the ore is placed. *C D E*, Condensing chambers. *G H*, Air channels.

chamber before entering another. In the last chamber (*D*) of each series, which is surmounted by the chimney, the last portions of mercury are condensed by a cascade of water. The

greater portion of the mercury is condensed into the liquid form in the first three chambers of each series, and is collected in an underground gutter which conveys it into a tank, from which it is ladled out to be filtered through cloth, and put into the wrought-iron bottles, containing about 60 lbs. each, in which it is imported into this country. The three last of each series of condensing-chambers receive the remainder of the mercury chiefly in the form of dust or soot, consisting of finely-divided mercury, with some sulphuret of mercury which has escaped in vapour, and some carbon from the bituminous matter in the ore. About a week is required to complete the distillation, including the time required for cooling the furnace. Only about $8\frac{1}{2}$ parts of mercury are obtained from 100 parts of ore. In 1803, a fire broke out in the mines at Idria, and the combustion was sustained by the bituminous matter in the ore, so that it became necessary to flood the workings with water. The vapours of mercury evolved proved very injurious to the health of the neighbourhood.

A process similar to that employed at Idria is followed at New Almaden, in California, for the extraction of the mercury.

Within the last few years, a greatly improved apparatus has been employed at Idria. The ore, in fragments as large as a fist, is thrown through a hopper into a deep cylindrical fire-place, upon which a little wood and coal are burnt. The mercurial vapours are conducted into six condensing chambers covered with iron plates and kept cool by a stream of water. The draught through the chambers is maintained by a chimney, where terraces are constructed over which water is allowed to flow. Fresh charges of 7 cwts. of ore and 28 lbs. of charcoal are introduced every three-quarters of an hour, and the spent ore is raked out from below through the moveable bars of the grate.

A still more scientifically constructed apparatus is employed at Idria for the treatment of small ores containing

one-hundredth or less of mercury, in which the ore is heated on the hearth of a reverberatory furnace (*a*, Fig. 124), the mercurial vapour being passed—1, through a condensing chamber (*d*); 2, through a wide sloping iron pipe (*f*) kept cool by the constant trickling of water over it, from the perforated gutter (*e*); 3, into a second condensing chamber (*g*); 4, through a second iron pipe (*h*), similar to the first, into a chimney (*k*). The hearth of the reverberatory furnace is divided into three compartments, the ore being first

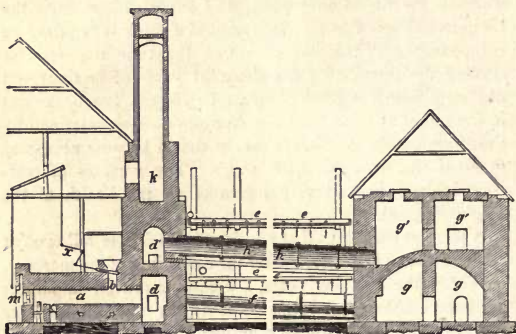


FIG. 124.—Modern Idrian Mercury-furnace.

thrown, through a hopper (*b*), upon that farthest from the grate, being raked out into each of the other compartments in succession, so as to be exposed to a gradually increasing temperature until it is drawn out in a spent condition through a channel near the fire-bridge, whilst fresh ore is charged at the other end of the hearth. Wood is the fuel employed. Beside the liquid mercury which is run out into proper receptacles, a large proportion of mercurial soot, containing finely-divided mercury, is collected in the condensing pipes and chambers. This is dried, raked over on

an inclined plane as long as any mercury runs out, and afterwards distilled to obtain the last portions of the metal.

On the west bank of the Rhine there are several small mercury mines which yield sulphuret of mercury associated with sandstone. These ores are made to yield their mercury by distilling them with lime, a process generally resorted to in the smaller mercury works. The distillation is effected in cast-iron pear-shaped vessels (A, Fig. 125), thirty of which are heated by the same fire, in a *gallery furnace* (M), the grate of which runs through its entire length, and is fed with coal,

which does not come into contact with the iron vessels, these being arranged in the upper part of the furnace, on each side of the grate, in two rows, one above the other, so that the flame may circulate around them before escaping through the openings into the chimney. The ground ore is mixed with about one-fourth of its weight

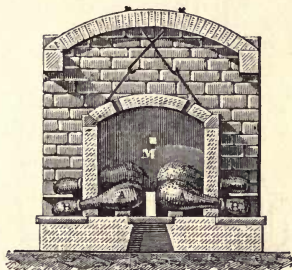


FIG. 125.—Extraction of Mercury in the Palatinate.

of quicklime, and about 70 lbs. of the mixture are introduced into each of the cast-iron bottles, which are then about two-thirds full. The neck of each bottle fits into a stone bottle (B) half full of water, placed outside the furnace, to receive the mercury. The water above the metal becomes filled with *black mercury* containing undecomposed sulphuret and finely-divided mercury; this is dried and distilled again with more lime.

In this process, the lime, or oxide of calcium, composed of calcium and oxygen, decomposes with the sulphuret of mercury, yielding sulphuret of calcium, which remains in the

iron bottle, mercury which passes over in vapour, and oxygen which converts a part of the sulphuret of calcium into sulphate of lime.

This operation in the gallery furnace is obviously attended with waste and inconvenience. The trouble of charging and discharging so many small bottles and of cementing the joints is very considerable, and has led to the introduction, in some places, of another arrangement which allows the mercury to be extracted on the same principle but with much greater economy.

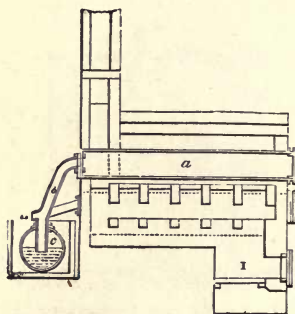


FIG. 126.—Ure's Retort for extraction of Mercury from Cinnabar.

In place of the bottles, cast-iron retorts (*a*, Fig. 126) are employed, resembling those used in distilling coal for gas. These are about seven feet long and one foot square in sectional area, so that they may be charged with 700lbs. (instead of 70) of the mixture of the ground cinnabar with lime, introduced at the back of the retort, which is

then closed with an iron plate. The front end is also closed with an iron plate (*a*, Fig. 127), which is provided with a sloping cast-iron pipe (*b*), 4 inches in diameter, having a door through which it may be cleared with a wire. This pipe dips into water contained in a condenser (*c*) resembling the *hydraulic main* of the gas works, being an iron pipe, 18 inches wide, and 20 feet long, which runs along the front of the range of nine retorts, and receives the mercury condensed from them, being itself kept cool by a stream of water running through a wooden trough around it. This pipe is a little inclined

towards one end, so that the condensed mercury may run down into a pipe (D) which conveys it into a locked cistern (e) (to prevent pilfering), in which an iron float (k) indicates the level of the mercury. The end of the pipe in the cistern is made to open into a small vessel, which is filled with mercury at the commencement to prevent the water from running out of the condenser. The latter is provided with a safety valve (g) to allow for any sudden expansion or contraction of the air within. The retorts are set like gas retorts, so that the same coal fire (I) may heat three of them, the range of nine having three furnaces with flues running into one chimney.

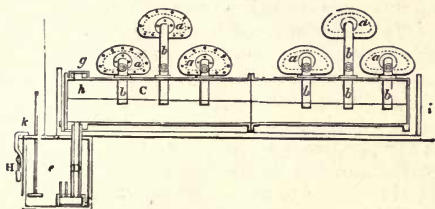


FIG. 127.—Ure's Retorts for extraction of Mercury.

In Hungary, a considerable quantity of mercury is extracted from the grey copper ore (*Fahlerz* or *fallow ore*) during the process of roasting preliminary to the smelting for copper. The roasting is effected by burning the ore in mounds about 40 feet by 20, and $4\frac{1}{2}$ feet high, in which channels are constructed for the admission of a moderate supply of air. A layer of small ore having been spread upon the ground, and covered with some larger pieces of ore already once roasted, some wood and coal are spread over them, then a layer of ore, the outside being covered with powdered ore so as to prevent a too free passage of air, and to retain the vapour of mercury. The heap is kindled through shafts left for the purpose, when the heat evolved by the

combustion of the wood and coal ignites the sulphur of the ore, which burns away as sulphurous acid, whilst the mercury is converted into vapour and condensed among the cooler portions of the fine ore on the outside of the heap. After about three weeks, the upper layers are thrown upon a sieve and washed with water, the fine ore which passes through the sieve being separated from the mercury by washing.

The mercury imported into this country generally contains lead, bismuth, and zinc as impurities, which cause globules of it to *tail* or leave a metallic streak behind them when rolled over a glass plate, whilst pure mercury runs off in globules, leaving the glass clean.

It is sometimes purified by re-distilling it, using as a retort one of the iron bottles in which it is imported; but the metal is liable to violent concussions during the ebullition, so that it is impossible to prevent a part of the impure mercury from splashing over into the receiver. A far better process for purifying it consists in pouring the mercury into a wide dish (such as a photographic tray), where it may form a thin layer, and covering its surface with diluted nitric acid (sp. gr. 1.15). The acid is allowed to remain in contact with it for a day or two, being frequently stirred, until the mercury no longer tails upon glass. The lead, bismuth, and zinc are dissolved by the acid, in the form of nitrates, together with a portion of the mercury, so that the acid should be preserved in order to assist in the purification of another portion of the metal. The mercury may be separated from the acid by pouring them both into a funnel stopped with the finger, on removing which the mercury runs off, the finger being replaced to retain the acid; the mercury is then well washed with water in a dish, and dried, first with blotting-paper, and afterwards at a gentle heat. Any dust or other mechanical impurities may be easily removed by filtering the mercury through a cone of writing-paper, in the apex of which a few pin-holes have been made.

An old and simple process for the purification of a small

quantity of mercury, consists in shaking it in a bottle with air and a little powdered sugar, which helps to divide the mercury and expose the lead, &c., to the action of the air, which converts them into a grey powder. By repeating the agitation with fresh portions of air, nearly all the foreign metals may be removed, and the sugar with the adhering oxides may be separated by filtering through pricked paper.

Many of the useful applications of mercury depend upon its being the heaviest substance which is liquid at the ordinary temperature. Its specific gravity being 13.54, a column of mercury thirty inches high serves, in the barometer, to measure the pressure of the atmosphere, whilst thirty-three feet (396 inches) of water are required for the same purpose.

The great interval between the temperature at which mercury congeals to the solid state (71 degrees below the freezing point of water) and that at which it boils (450 degrees above the boiling-point of water) renders it especially suitable for filling thermometers, an application for which it is also recommended by the circumstance that it has a very low *specific heat*, or requires much less heat to raise it to a given temperature than most other liquids which could be employed for thermometric purposes : thus, a spirit thermometer, in which alcohol is used to show the expansion, rises and falls much more slowly than the mercurial thermometer, because the specific heat of alcohol is 15 times as great as that of mercury ; in other words, fifteen seconds would be required by a spirit-thermometer to measure a temperature which would be indicated in one second by a mercurial thermometer of the same weight. A further advantage on the side of mercury is derived from its not adhering to the glass. Moreover, in a spirit-thermometer, the indications are affected by the presence of the vapour of alcohol in the space above the column, which should be perfectly vacuum, and is nearly so in the case of the mercurial thermometer, for the metal does not evolve an appreciable amount of vapour at temperatures below the boiling-point of water.

That mercury does give off a minute quantity of vapour, even at the ordinary temperature, is shown by the appearance of minute globules of metal condensed, in cold weather, upon the glass in the upper part of a barometer, and by the experiment of suspending a gold leaf at an inch or two above the surface of mercury in a bottle, when it is found, after some time, to be whitened by the combination of mercury vapour with the gold.

Mercury is remarkable for its property of uniting with other metals at the ordinary temperature, to form combinations which are termed amalgams. Iron and platinum are the only metals in ordinary use which are not attacked by mercury. It adheres to platinum and wets its surface, but does not unite chemically with it. Gold is soon penetrated by mercury, and becomes very brittle. The surface of a gold ring, for instance, is instantly whitened by mercury, and if allowed to remain in contact with it for a short time, the gold is rendered so brittle as to be useless; a mere external coating of amalgam may be removed, and the colour of the gold restored, by warming it with a little nitric acid, the surface of the gold being afterwards burnished.

Silvering Looking-glasses.—An amalgam of tin is employed for silvering the backs of looking-glasses, which is performed in the following manner. A sheet of tin-foil (generally hardened by a minute proportion of copper), somewhat larger than the plate to be silvered, is laid upon a stone or marble table, which is swung upon an axis so that it may be gradually sloped by a screw when required. The upper surface of this table is perfectly smooth and level, and has a gutter running round it, with a spout for collecting the superfluous mercury. The tin-foil is applied to the table with a brush, so that it may be free from wrinkles, and the surface having been laid truly horizontal, a little mercury is spread over it with a roll of flannel, so that every part of the tin-foil may be amalgamated. A very thin layer of mercury is then poured over the surface, and the edge of the

clean dry plate to be silvered is very carefully pushed forward over the table, so as to carry the superfluous mercury before it, and to prevent any air from entering between the amalgam and the glass. Some flannel is placed on the glass, with a weight upon it, and the table is very slightly inclined to drain off the excess of mercury. After about five minutes, the plate is loaded with several heavy weights, and allowed to remain for twenty-four hours, in order that the amalgam may be made to adhere firmly to the glass; the inclination of the table is somewhat increased from time to time, to promote the draining away of the excess of mercury. When the weights are removed, the plate is laid upon a sloping wooden table, the upper edge of which is raised gradually by a pulley until the plate is perpendicular. After three or four weeks, when the excess of mercury has drained off, the looking-glass is ready for framing. The amalgam adhering to the glass contains one part of mercury and four parts of tin.

The amalgam which is employed to promote the action of glass electrical machines is composed of two parts of zinc and five parts of mercury.

Magnetic amalgam is the somewhat fanciful name bestowed upon the amalgam of one part of metallic sodium with thirty parts of mercury, which is liquid at a very moderate heat, but solidifies on cooling to a hard crystalline mass. It is cast into ingots which are kept in air-tight iron vessels with lime, to absorb any moisture, which would act upon the sodium in the amalgam and convert it into soda. This amalgam is exported to the amalgamating mills for gold and silver ores, where it encourages the amalgamation and prevents *flouring* (p. 349).

An amalgam of one part of cadmium and three parts of mercury is employed by dentists.

PLATINUM.

This valuable metal has been brought into use in quite modern times, having been discovered by an assayer in Jamaica in 1741, becoming generally known in Europe in 1748. Its name, derived from the Spanish, signifies *little silver*, since it somewhat resembles that metal in colour; it has also been called *white gold*, and it is said that when first discovered, much of it was thrown away, lest, from its durability and high specific gravity, it should be employed for debasing gold, an expectation which has been partly realised (p. 381).

Platinum always occurs, like gold, in the metallic state, and most commonly in alluvial deposits in which gold is also present. Nuggets of platinum have rarely been found, the largest on record being one of 18 lbs. weight. The metal is commonly met with in flattened grains of a light steel-grey colour. The Ural mountains have furnished the largest quantity of platinum, but it has also been obtained from Brazil, Peru, Borneo, Australia and California.

The *Platinum ore*, as it is called, is separated from the earthy matters by washing the latter away, when the grains of platinum remain behind with grains of gold, magnetic iron ore, corundum, and a very heavy alloy of osmium and iridium. The platinum itself is far from pure, containing only from 75 to 85 parts of that metal in a hundred, the remainder consisting of iron, sometimes found to the amount of 13 per cent., iridium, rhodium, palladium, osmium, and copper. When any considerable quantity of gold dust is present, it is separated from the platinum by the process of amalgamation. The magnetic iron ore is extracted by a magnet.

The original process for extracting platinum from the ore is rather a chemical than a metallurgical operation, but since

it was, until within the last few years, the only method by which the metal was produced in a marketable state, a short description of it is here given.

The grains of platinum ore are heated with nitric acid, which dissolves any silver, copper, iron, and lead, in the form of nitrates; after these have been extracted, the residue is washed with water, and heated with hydrochloric acid, which dissolves the magnetic oxide of iron; it is then again washed with water, and gently heated for several hours with hydrochloric acid to which a little nitric acid is added from time to time. The platinum is thus dissolved, together with palladium, rhodium, and some iridium, whilst the osmium and the rest of the iridium are left undissolved. The acid solution containing the chlorides of platinum, &c., is poured off, and the residue heated with fresh portions of acid as long as anything is dissolved, when any quartz and corundum are left, together with the grains of the alloy of osmium and iridium, which are employed, on account of their surpassing all other metallic substances in hardness, for making the nibs of gold pens upon the points of which they are soldered.

The liquid containing the chloride of platinum is mixed with a solution of sal-ammoniac (muriate of ammonia) containing one-sixth of its weight of the salt, of which about four parts are employed for every ten parts of the ore. A yellow precipitate is then deposited, which contains the greater part of the platinum, in the form of *ammonio-chloride of platinum*, a combination of sal-ammoniac with chloride of platinum.

This yellow precipitate is washed with cold water, dried, and strongly heated in a plumbago crucible, when the sal-ammoniac and the chlorine are driven off, and *spongy platinum* is left as a grey porous mass. This is finely powdered in a wooden mortar, rubbed to a paste with water, passed through a sieve in order to render it perfectly uniform, and poured into a slightly conical brass mould closed below with blotting-paper wrapped round a steel stopper. When the

water has drained off, a plunger is forced in by a coining press, so as to condense the mass, which has at first the specific gravity 4.3, until its specific gravity is 10, when it has been reduced to about two-fifths of its former bulk, and has acquired a metallic appearance. The disk is now sufficiently coherent to be removed from the mould and intensely heated

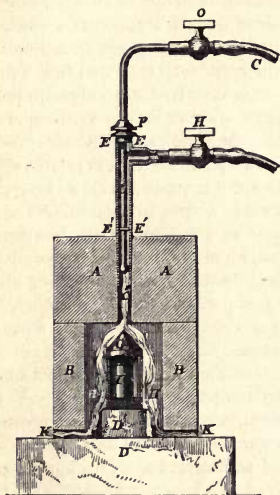


Fig. 128.—Platinum melted in Crucible by the Oxy-hydrogen Blowpipe.

for about 36 hours in a porcelain kiln, when it contracts to about four-fifths of its former volume. It is taken out of the furnace at a white heat, and hammered upon its ends, not upon the sides, lest it should crack. After being heated and hammered in this way several times, the particles become thoroughly welded together into a compact malleable mass of metal, of specific gravity 21.5.

Instead of welding the platinum into a compact mass, it is now sometimes melted in a crucible by the intense heat of the oxy-hydrogen blowpipe-flame. The

crucible is made of gas-carbon, and is enclosed in another crucible (1, Fig. 128) made of lime, and provided with a conical cover (G) of the same material. The furnace is made of three blocks of lime strengthened by iron wire, one (D) serving for the hearth, upon which is placed a hollow cylinder (B) of lime, which has been bored in the lathe until it is wide

enough to leave a clear space (H) of about $\frac{1}{8}$ th inch round the lime crucible. At the lower part of this cylinder are four openings (K) for the escape of the steam produced in the combustion of the hydrogen and oxygen. The furnace is covered in with a block of lime (A) about $2\frac{1}{2}$ inches thick, in which a slightly conical passage is bored for the reception of the blowpipe-jet which passes down to within an inch of the apex of the conical cover of the lime crucible. The blowpipe (E) consists of two concentric copper tubes, the outer one through which the hydrogen is passed being about $\frac{1}{2}$ inch in diameter, and terminating in a somewhat tapering nozzle of platinum (C) about $1\frac{1}{2}$ inch long, which fits into the conical passage through the upper block of lime. The oxygen is conveyed through the inner copper tube, also furnished with a platinum nozzle, the opening of which is about $\frac{1}{10}$ th inch in diameter.

The hydrogen and oxygen are supplied from their respective gas-holders, their passage being regulated by the stop-cocks H O, under a pressure of about 16 inches of water, the hydrogen being lighted before the oxygen is turned on.

The new process for extracting platinum from its ores resembles one of the methods of extracting gold and silver, the metal being dissolved out by melted lead and afterwards recovered by cupellation.

Two hundredweight of the platinum ore mixed with an equal weight of galena (sulphuret of lead) is thrown, in small portions, into the concave hearth of a small reverberatory furnace built of fire-brick. The materials are stirred with an iron rod until the platinum has entirely dissolved in the fused galena. A little glass is then introduced, to melt over the surface, and a quantity of litharge (oxide of lead) equal in weight to the galena is gradually added. The sulphur in the sulphuret combines with the oxygen of the oxide, and passes off as sulphurous acid gas, leaving the metallic lead in combination with the platinum, whilst the alloy of osmium and iridium, being unaffected by the lead, sinks, in separate grains, to the

bottom (its specific gravity varying, according to its composition, from 19·4 to 21·1). After remaining at rest for some time, the upper portions are ladled out and cast into ingots, and the remainder is added to the next charge.

The lead containing platinum is treated in a cupellation furnace (p. 336), when the lead is removed as an oxide,

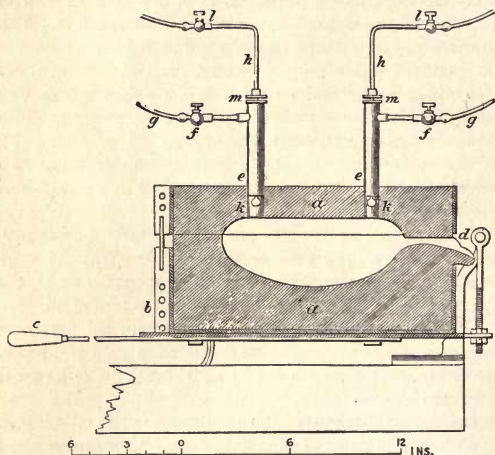


FIG. 129.—Lime Furnace for melting Platinum with the Oxy-hydrogen Blow-pipe. *a a*, Blocks of lime hollowed out to form the furnace. *h h*, Openings for the oxy-hydrogen blowpipes. *e e*, Outer tubes conveying the hydrogen or coal-gas. *h h*, Inner tubes conveying the oxygen. *d*, Spout for pouring the melted platinum.

leaving the platinum in a spongy state upon the cupel, whence it is transferred to a small furnace made of lime (Fig. 129) and melted by the flame of the oxy-hydrogen blowpipe (in which coal-gas may be substituted for hydrogen), the intense heat of which volatilises any silver, gold, lead, palladium and osmium. When the metal is sufficiently

refined, it is poured through an opening in the side of the furnace into an ingot mould made of gas-carbon or of wrought iron lined with platinum. In this manner, 25 lbs. of platinum have been melted and refined in three-quarters of an hour, with a consumption of about 43 cubic feet of oxygen. The melted platinum resembles silver in its property of absorbing oxygen mechanically at a high temperature and evolving it again as it cools, exhibiting the phenomenon of *sprouting* (p. 339).

Since platinum is one of the most malleable and ductile of the metals, being surpassed, in the former quality, only by gold, silver and copper, and in the latter by gold and silver, it is easily rolled into sheets or drawn into wire.

The principal uses of platinum depend upon its resistance to the action of heat, of oxygen, and of acids. The largest quantity of the metal is devoted to the manufacture of the stills employed for boiling down oil of vitriol in order to expel the water, and of the siphons used for drawing the hot acid out of the stills. Similar stills are also employed in the operation of parting gold and silver with sulphuric acid (p. 375). The joints of these stills are soldered with fine gold, and they are usually gilded inside, for otherwise they are liable to become porous under the influence of the boiling acid, allowing it to exude. They are protected from external injury and from the direct action of the fuel by an iron casing. Some platinum stills have been made weighing upwards of 60 lbs. and costing above 2,000*l*. They are slowly corroded by the action of the acid, and require occasional repair by soldering them with gold.

Platinum is also much employed for evaporating basins and crucibles for chemical purposes, and thin foil and wire of this metal are indispensable in operations with the blowpipe. When the particles of the metal have been imperfectly consolidated by hammering, it is found to blister under the influence of a very high temperature. The permanence of platinum under the action of heat has led some persons to

adopt an erroneous estimate of its durability under other conditions, so that it is often forgotten that platinum is a soft metal and therefore ill-adapted to resist ordinary wear. It is easily corroded and rendered brittle by carbon and silica, both of which are present in coal, coke, and charcoal, for which reason platinum crucibles are never allowed to come into direct contact with the solid fuel, but are heated either in the flame of a gas or spirit lamp, or in a muffle (p. 383), or enclosed in a clay crucible lined with magnesia to prevent the platinum from sticking to the heated clay.

Metals must never be melted in platinum crucibles, since most of the metals are capable of forming alloys with it. Caustic alkalies and saltpetre in a melted state also act upon the metal, and phosphorus and arsenic combine with and corrode it very rapidly at moderately high temperatures.

Neither sulphuric, hydrochloric, nitric or hydrofluoric acid separately has any action upon platinum, but a mixture of hydrochloric with nitric acid dissolves it, though more slowly than it dissolves gold.

An alloy of platinum, iridium and rhodium is sometimes employed for crucibles, which are harder and less easily corroded than those made of pure platinum. To obtain the alloy, the ore of platinum (which contains the two other metals) is mixed with a quantity of lime equal to that of the iron contained in the ore, and fused by the oxy-hydrogen blowpipe in a furnace made of lime (p. 408). The iron and copper are converted into oxides which form a fusible slag with the lime, whilst the gold, palladium and osmium are expelled in the form of vapour, and the alloy of platinum, iridium and rhodium remains.

Small tubes, &c., may be easily extemporised with platinum wire and foil, by taking advantage of the readiness with which surfaces of this metal unite when hammered at a high temperature.

Platinum vessels are cleaned by smearing them with a paste containing equal bulks of borax and cream of tartar

with a little water, drying and heating them till the mixture melts, and immersing them for several hours in diluted sulphuric acid. Heating in contact with fused bisulphate of potash, or with powdered sal-ammoniac, is also employed for the same purpose. The platinum vessels are finally well washed with water and burnished with agate.

Platinum is sometimes employed for the touch-holes of small-arms, and for the vents of cannon, on account of its resistance to corrosion. The circumstance that it expands less than any other metal when heated, enables it to be cemented into glass, by fusing the latter, whilst other metals which differ much from glass in their rate of expansion by heat, would crack it as they cool. This renders platinum of great importance in the fabrication of various philosophical instruments.

Though pure platinum is unaffected by nitric acid, it may be rendered soluble in that acid by previously alloying it with ten or twelve times its weight of silver, which is taken advantage of in order to separate platinum from gold in the process of assaying the latter metal with which platinum is frequently associated. If the platinum be present in small proportion (not exceeding 3 or 4 per cent.) in the alloy of gold and silver obtained by cupellation (p. 383), the whole of it will be dissolved together with the silver, in parting by nitric acid; but when the quantity of platinum is larger, which is indicated by the difficult fusibility of the button on the cupel, and by the blanched appearance of the gold eventually obtained, the latter must be again fused with at least three times its weight of pure silver, the alloy rolled very thin, and boiled for half an hour, and a quarter of an hour, respectively, with the two strengths of nitric acid mentioned at page 383, in order to remove the whole of the silver and platinum. An alloy of silver with one-third of its weight of platinum is employed by dentists on account of its great elasticity.

The remarkable property of platinum, especially in the

finely-divided states of spongy platinum and platinum black, to condense gases into its pores and thus to promote their chemical action upon each other, is not suited for description in a metallurgic treatise.

PALLADIUM is generally found in small quantity, not exceeding 1 per cent., associated with the ore of platinum, from which it is extracted by a process which is purely chemical. Formerly there existed a pretty abundant source of this metal in the form of an alloy with gold found in the mines of Brazil, but of late years this has failed, and palladium has risen to an extremely high price. In appearance it resembles platinum, but is much harder, though it possesses considerable malleability and ductility. It is quite unchanged by air at the ordinary temperature, but assumes a bluish colour, from the formation of a thin film of oxide, at a moderately high temperature, becoming bright again at a higher temperature, the oxide being decomposed. Palladium fuses at a somewhat lower temperature than platinum, but cannot be fused in a furnace. It is only half as heavy as platinum, its specific gravity being 11.5, so that it is much better adapted for making very accurate balances and other philosophical apparatus. The graduated scales of astronomical instruments are often made of palladium, and an alloy of this metal with $\frac{1}{10}$ th of silver has been sometimes employed by dentists.

ANTIMONY.

Though antimony is far too brittle to be employed in its pure state for any useful purpose, it has been shown to be of great service in hardening the softer metals lead and tin, so that the history of this metal is not devoid of interest for the metallurgist.

Antimony is occasionally found in Nature in the metallic state, as at Andreasberg in the Hartz, where it is alloyed with small quantities of silver, iron, and arsenic. The only ore from which it is largely extracted is the *grey antimony ore*, a sulphuret of antimony, containing, when pure, $71\frac{3}{4}$ parts of antimony combined with $28\frac{1}{4}$ parts of sulphur. It is found in Cornwall, Auvergne, Hungary and Borneo, associated with galena and iron pyrites, and with quartz and heavy spar, in veins traversing rocks of granite or slate. The appearance of grey antimony ore is very characteristic; it commonly resembles a compact bundle of dark grey metallic needles converging towards one point, and often exhibiting a blue iridescence due to a thin film of oxide. It is very heavy (sp. gr. 4.63), and melts easily even in the flame of a candle. This fusibility is taken advantage of in order to separate it from the earthy matters, to effect which the ore is heated on the concave hearth of a reverberatory furnace, the hearth being lined with charcoal to prevent oxidation of the sulphuret, which melts and is run out into moulds, where it is cast into the form of the cakes sent into commerce as *crude antimony*, which contains, in addition to the sulphuret of antimony, sulphurets of arsenic, iron and lead.

The *oxide of antimony* occurs in Algeria, and is smelted in France.

Red antimony, a compound of oxide and sulphuret of antimony, is found in Tuscany, and smelted at Marseilles.

Regulus of antimony, or metallic antimony, is extracted from the sulphuret by melting it upon the hearth of a reverberatory furnace in contact with metallic iron (clippings from the tin-plate works), which removes the sulphur, forming sulphuret of iron; this collects above the melted antimony, which is run out into moulds. It contains a considerable quantity of iron.

Sometimes the regulus is extracted directly from the rich antimony ore, without previous production of crude antimony. For this purpose the ore is broken into pieces as large as an egg, and introduced into red-hot crucibles, together with a little alkaline slag; some scrap-iron is placed on the top and pressed down when the mass has fused. After about two hours, the melted antimony and sulphuret of iron are poured into conical iron moulds, where they separate into two layers.

A purer metal is obtained by the following process. The sulphuret of antimony, or the rich original ore, is crushed, and roasted, without being melted, for six hours, in a reverberatory furnace, when most of the sulphur is expelled as sulphurous acid gas, and most of the arsenic as arsenious acid, whilst part of the antimony is converted into vapour, and combines with oxygen to form the oxide of antimony, which is carried into the flues of the furnace.

The roasted ore, which has a red-brown colour, contains the oxide and sulphuret of antimony. It is ground to powder, mixed with about one-fifth of its weight of charcoal, some chloride of sodium, carbonate of soda, sulphate of soda, and slags from a former operation. The mixture is thrown upon the hearth of a reverberatory furnace, and well stirred, when the oxide of antimony in the roasted ore is reduced to the metallic state by the charcoal, whilst the sulphuret of antimony exchanges its sulphur for the oxygen of the soda, yielding oxide of antimony, which is also reduced by the charcoal, and sulphuret of sodium, which forms a slag with the sulphurets of other metals present, and with the

chloride of sodium. The metal and slag are run off into an outer basin.

The fumes of oxide of antimony are condensed in long flues. The poorer ores, after being roasted, are smelted in cupola furnaces with coke.

The antimony is refined by melting it, in quantities of 60 or 70 lbs., with 1 or 2 lbs. of American potashes (carbonate of potash) and 10 lbs. of the slag. It is then allowed to solidify quietly under a layer of slag, in order that it may assume the beautiful fern-like crystalline markings on its surface which have gained for it the name of *star-antimony*.

The above process for extracting the metal is far from economical, little more than one-half of the antimony present being obtained in the metallic state.

Antimony can easily be distinguished from every other metal by its hardness, brittleness and crystalline structure ; a slight tap with a hammer suffices to break an ingot of antimony, and the broken surface exhibits large shining plates ;* it is so brittle that it may be easily reduced to a fine powder in a mortar. It is comparatively a light metal, its specific gravity being only 6.7. It melts at 800° F., and at a higher temperature it gives off much vapour, which produces a thick white smoke of oxide of antimony. Its applications have been noticed in the preceding pages.

* The addition of a minute proportion of tin to antimony causes it to crystallise more readily and in larger crystals.

BISMUTH.

Bismuth, or *marcasite** as it was formerly termed, is a comparatively rare metal which is found associated with the ores of nickel, cobalt, copper and silver, chiefly in Saxony, Transylvania and Bohemia. It also occurs in smaller quantity in Cornwall, Cumberland, Stirlingshire, Norway, Sweden, and the United States, and has lately been found in Peru. Bismuth is always extracted from the ores which contain it

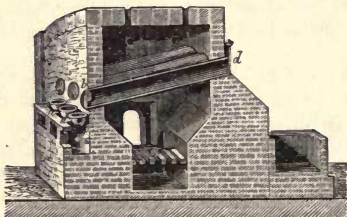


FIG. 130.—Extraction of Bismuth.

in the uncombined metallic state, by taking advantage of the readiness with which it fuses (507° F.) and drains away from the other constituents of the ore. It is extracted chiefly at Schneeberg in Saxony, from an ore containing from seven to twelve parts of bismuth in a hundred, associated with a compound of arsenic and cobalt. This is broken into pieces about the size of a nut, and introduced, in charges of about 50 lbs., into sloping cast-iron cylinders (Fig. 130) heated by a wood fire. The lower opening (*b*) of each cylinder is closed with a fire-clay stopper having an aperture through which the melted bismuth may run out. The upper open-

* A term sometimes applied also to iron pyrites.

ings are closed by an iron door (*d*); in about ten minutes after charging, the metal begins to run into an iron pot (*a*), kept hot by a separate fire in the channel (*κ*), and containing a little coal-dust to prevent the oxidation of the bismuth. The ore is raked about occasionally, to promote the separation of the metal, which is exhausted in the course of half an hour, when the residue is raked out from the upper end of the cylinder, where it falls, down an incline, into a trough of water, and a fresh charge is introduced. With five cylinders over a single furnace, one ton of ore is smelted in eight hours. The iron pans are emptied into moulds in which the bismuth is cast into bars weighing from 25 to 50 lbs. each.

At Joachimsthal, the bismuth ores have of late been smelted in crucibles, by melting them with about a fourth of their weight of scrap-iron (to combine with the sulphur), half their weight of carbonate of soda, to convert the silica into a slag of silicate of soda, one-twentieth of lime and the same weight of fluor spar. One hundredweight of the mixture is introduced into each crucible, and a part of the carbonate of soda is employed to cover the mixture. The crucible is closed with a lid, and strongly heated in a furnace. When the mass is pasty it is well stirred in order to effect perfect mixture, and, after complete fusion, it is ladled out into conical iron moulds, when the bismuth subsides into the narrower part of the mould.

As it comes into the market, bismuth contains considerable quantities of arsenic, iron and silver which do not, however, seriously interfere with its limited applications. A great part of the arsenic is sometimes expelled by heating the metal in crucibles, filled up with charcoal to prevent access of oxygen, when the arsenic passes off in vapour.

When a sufficient quantity of silver is present, it is extracted from the bismuth by the process of cupellation (p. 336), the bismuth becoming converted into an oxide, which is removed, like the litharge formed when lead is

cupelled, leaving the silver upon the cupel. It is stated that the great rise in the price of bismuth during the last few years is partly due to the circumstance that large quantities of it have been bought up and cupelled for the sake of the silver which it contains. When lead containing bismuth is cupelled, the lead is first converted into oxide, so that towards the end of the process, nearly pure bismuth is left on the cupel, which will oxidise in its turn if the process be continued.

Bismuth is known by its peculiar reddish colour and its highly crystalline appearance. A blow with the hammer breaks it easily, though it is not quite so brittle as antimony. Its tendency to crystallise is very remarkable; by melting some bismuth in a ladle, allowing a solid crust to form upon the surface, piercing this with two holes, and pouring the liquid bismuth out of one of them (the other allowing the air to enter), a mass of beautiful cubical crystals of bismuth is obtained.

Its brittleness renders bismuth unfit for use in the metallic state, by itself, except for the construction of *thermo-electric piles*, which are made of alternate bars of bismuth and antimony, and are employed as very delicate thermometers, in order to measure slight differences of temperature by the electric currents which they produce.

The uses of bismuth in alloys depend upon its low melting-point (507° F.), and its property of expanding very considerably during solidification, the solid metal occupying $\frac{1}{32}$ nd more space than the liquid metal, affording an exception to the general law, which requires that cooling should produce contraction. Another remarkable physical peculiarity of bismuth is the circumstance that the specific gravity of the metal is diminished instead of being increased by strong pressure. Thus, a cylinder of bismuth of specific gravity 9.783 was placed in a steel cylinder fitted with a plunger, and subjected to a pressure of 200,000 lbs., when its specific gravity was found to have diminished to 9.556,

having been reduced by about $\frac{1}{3}$ rd part. The compressed bismuth exhibited scarcely any crystalline structure.

Newton's fusible alloy is composed of two parts of bismuth, one of lead, and one of tin, and melts at 201° F., so that it liquefies readily in boiling water, although the most easily fusible of its constituents, the tin, has a melting-point of 442° F. Such an alloy is used as a soft solder by pewterers.

Some kinds of type-metal and stereotype-metal contain bismuth in order that they may expand into the finest lines of the mould during solidification. For a similar reason, an alloy of tin, lead, and bismuth is employed for testing the finish of a die.

Bismuth is more easily converted into vapour than many other metals, and may be boiled at a moderate white heat.

ALUMINUM.

This metal, which is now often called *Aluminium*, although discovered by Wöhler in 1828, has only within the last few years been found capable of useful application in its metallic form. Though never found as a metal in Nature, it is probably the most abundant of all metals in a state of combination, since it exists in every variety of clay (silicate of alumina), its quantity varying from twelve to twenty parts in a hundred. Another mineral containing aluminum is *kryolite*,* in which the metal is combined with sodium and fluorine, and forms 13 per cent. of the mineral, which is found in abundance in Greenland.

* So called from the Greek for *frost* on account of its resemblance to ice.

Aluminum is extracted from a particular variety of clay known as *bauxite*, which is found at Baux, near Arles, in the south of France; this mineral contains about one-third of its weight of aluminum, combined with oxygen (forming *alumina*), together with silica, oxide of iron, and water. At Newcastle,* where the metal is extracted from bauxite, the following process is adopted:

The ground mineral is mixed with *soda-ash* (containing carbonate of soda and caustic soda) and heated in a reverberatory furnace, when the soda combines with the silica and alumina, forming compounds known as *silicate of soda* and *aluminate of soda*, whilst the carbonic acid is expelled in the form of gas. The mass, after cooling, is treated with water, which dissolves the aluminate of soda. This solution is mixed with enough hydrochloric (muriatic) acid to remove the soda, when the alumina is separated as a gelatinous precipitate composed of *hydrate of alumina*, a compound of alumina with water. This is mixed with common salt (chloride of sodium) and charcoal powder, to a stiff paste, which is made up into balls as large as an orange, very thoroughly dried, and strongly heated in earthen cylinders through which perfectly dry chlorine gas is passed.

The carbon of the charcoal combines with the oxygen of the alumina, escaping as carbonic oxide gas, whilst the aluminum unites with the chlorine to form the chloride of aluminum; the latter enters into combination with the chloride of sodium, producing a *double chloride of aluminum and sodium* which distils over and condenses to a solid salt. Ten parts of this salt are mixed with two parts of sodium in small pieces, and with five parts of kryolite or of fluor spar, to form a liquid slag which shall cover the surface of the metal. This mixture is thrown upon the red hot hearth of a reverberatory furnace, which is then immediately closed to exclude air. The sodium acts violently upon the chloride

* Aluminium is no longer manufactured at Newcastle.

of aluminum, abstracting its chlorine and liberating the aluminum, which collects, in the melted state, beneath a layer of slag containing the chloride of sodium and kryolite. The metal thus obtained always contains silicon and iron in considerable quantity.

Aluminum is a white malleable metal about as hard as zinc, and fusing at a somewhat lower temperature than silver. It is remarkably light, having a specific gravity of only 2.5, and is unaffected by air; unlike silver, it is not even tarnished by air containing sulphuretted hydrogen. A bar of aluminum suspended from a string sounds like a bell when lightly struck. In manufacturing objects of ornament from aluminum, a solder is employed which contains ninety parts of zinc, six parts of aluminum and four parts of copper.

At present, the principal demand for aluminum in this country is for the manufacture of *aluminum-bronze* or *aluminum-gold*, which is an alloy of aluminum with nine times its weight of copper (see p. 294).

An alloy of silver with two-thirds of its weight of aluminum is used in France, under the name of *tiers-argent*, as a substitute for silver, being much harder than that metal and less than half the price.

Aluminum is sometimes employed for making small weights, for which it is well adapted by its lightness and resistance to the action of air. The beams of small balances have also been made of aluminum.

MAGNESIUM.

Like aluminum, this metal has only been extracted in any quantity during the last few years, a considerable demand for it having arisen in consequence of its property of burning with a very brilliant white light which is found useful for the illumination of microscopes, magic lanterns, &c., as well as for taking photographs at night or in places where daylight does not penetrate.

Magnesium occurs abundantly, in combination with oxygen and carbonic acid, in *magnesite* (carbonate of magnesia) and *dolomite* or magnesian limestone (carbonate of lime and magnesia). Another source of the metal is the recently-discovered mineral *carnallite*, which is found in large quantity above the rock-salt in the salt mines of Stassfurth in Saxony. This mineral is composed of magnesium, potassium, chlorine, and water, and contains about one-twelfth of its weight of magnesium. The water may be expelled by heat, leaving the *double chloride of magnesium and potassium*.

Magnesium may be extracted from the dried carnallite by mixing it with one-tenth of its weight of fluor spar, to act as a flux, and one-tenth of its weight of sodium in small pieces. By fusing this at a moderate heat, the chloride of magnesium is made to give up its chlorine to the sodium, and the magnesium collects in the melted state beneath a liquid slag composed of chloride of sodium, chloride of potassium, and fluoride of calcium. The magnesium may be purified by distilling it, in an iron crucible, as practised in the case of zinc (p. 276). Magnesium bears considerable resemblance to aluminum, but is a whiter metal, and even lighter than

aluminum, its specific gravity being only 1.74. It may be liquefied below a red heat, and, as stated above, may be readily distilled. It is a little more tarnished than zinc when exposed to air. The magnesium wire is made by forcing the heated metal through holes in a steel plate, and magnesium riband, by passing the wire between heated rollers. When the end of a piece of wire or riband is held in a flame, it catches fire and burns with a dazzling light, the magnesium combining with the oxygen of the air to form a white earthy mass of magnesia.

The *sodium* required for the extraction of aluminum and magnesium is extracted directly from carbonate of soda, which is itself made from common salt (chloride of sodium). The well-dried carbonate of soda is mixed with powdered charcoal, some chalk being added to prevent the fusion of the mixture, which is strongly heated in wrought-iron cylinders protected from the fire by a coating of clay. The carbonate of soda contains sodium, oxygen, and carbonic acid; the carbon of the charcoal combines with the oxygen, and the sodium is converted into vapour and condensed in vessels containing petroleum; for sodium cannot be exposed to the air, even for a few minutes, without combining extensively with oxygen, and it even takes up that element, with great violence, from water, in which the oxygen is united with hydrogen. Sodium would scarcely be taken for a metal by an ordinary observer, in the state in which it is found in commerce, where it occurs in greyish earthy-looking light masses; but when these are cut with a knife, the fresh surfaces exhibit a brilliant lustre.

CADMIUM.*

Cadmium is found, in small quantities, not exceeding 2 or 3 per cent., in the ores of zinc, and distils over, together with the first portions of zinc, during the smelting of the ores of that metal, the period of its distillation being known as the *brown blaze*, because its vapour imparts a brown colour to the flame. If these first portions of metal, mixed with oxide, be mixed with charcoal and distilled again, the first portions, being collected apart, will contain a still larger proportion of cadmium, this metal being much more easily converted into vapour than zinc is. To obtain pure cadmium, the mixture of zinc and cadmium is dissolved in diluted sulphuric acid, and sulphuretted hydrogen gas is conducted into the solution, when the sulphur combines with the cadmium to form a bright yellow sulphuret of cadmium which is deposited. This is washed, dissolved in strong hydrochloric acid, and converted into carbonate of cadmium by adding carbonate of ammonia. The carbonate of cadmium being washed, dried, and distilled with charcoal, as in the case of zinc, yields metallic cadmium in a pure state.

Cadmium resembles tin in colour and appearance, as well as in the property of *creaking* when bent. It is a malleable and ductile metal at the ordinary temperature, but becomes brittle at about 180° F. It melts at the remarkably low temperature of 442° F., the melting-point of tin, and an alloy of three parts of cadmium, fifteen of bismuth, eight of lead, and four of tin fuses at 140° F.

Cadmium is harder than tin, and possesses greater tenacity. It is also somewhat heavier, its specific gravity being 8.6.

* From *cadmia* (Latin), *brass-ore*, referring to its connexion with the ores of zinc.

TABLES.

TABLE I., showing the Names and some of the Properties of the more important Metals.

Name of Metal	Sym- bol	Atomic Weight	Specific Gravity	Fusing Point approx. in degr. Fahr.	Length at 212° F. of a bar which measures 1 at 32° F. §	Weight of one cub. ft. in lbs.	Tensile Strength per sq. in. in tons
Aluminium*	Al	27	2.67	1292°	1.0022	166.8	12
Antimony .	Sb	122	6.72	1150°	1.0011	419.5	0.5
Bismuth .	Bi	210	9.82	507°	1.0014	613.0	1.5
Cadmium*	Cd	112	8.69	442°	1.0031	542.5	—
Cobalt* .	Co	59	8.95	< Iron	—	558.7	= Iron
Copper* .	Cu	63.5	8.94	1996°	1.0017	558.1	13 to 15
Gold* .	Au	196.6	19.36	2016°	1.0015	1208.6	9.1
Iron* .	Fe	56	7.84†	3500°	1.0012	489.4	29.0†
Lead .	Pb	207	11.36	617°	1.0028	709.2	8 to 15*
Magnesium	Mg	24.3	1.74	850°	1.0028	108.6	—
Manganese	Mn	55	8.01	< Iron	—	500.0	—
Mercury .	Hg	200	13.59	— 39°	1.0180‡	848.4	—
Nickel* .	Ni	59	8.67	< Iron	—	541.2	= Iron
Palladium*	Pd	106.5	11.8	= "	1.0010	736.6	—
Platinum* .	Pt	197.1	21.53	> "	1.0009	1344.0	—
Silver .	Ag	108	10.53	1873°	1.0021	657.3	18.2
Tin .	Sn	118	7.29	442°	1.0023	455.1	2.0 to 3.5*
Zinc* .	Zn	65	7.14	773°	1.0029	445.7	3.3 to 8.3*

NOTE.—(1) The figures in the tables must be looked upon only as approximately true : no two pieces of metal would give precisely the same result. (2) < is used to signify *less than*, = *equal to*, and > *greater than*.

* Signifies that the figures refer to wrought metal.

† Maximum. The minima are sp. gr. 7.5, tensile strength 16.

‡ Total expansion.

§ For the same kind of metal, the greater its density, the greater its expansion for a given increase in temperature.

TABLE II., showing the Specific Gravity, &c. of some of the Non-metallic Materials employed in works.

Name of Substance	Sp. Gr.	Weight of one cub. ft. in lbs.	No. of cub. ft. in one ton	Crushing strain in lbs. per sq. in.
Brickwork, common . {	1·6 to 2·0	100 to 125	22·4 to 17·9	—
„ London stock .	1·84	115	19·4	—
„ red . . .	2·16	134	16·7	808
„ Stourbridge fire	2·2	137	16·3	1,717
„ Welsh fire .	2·4	150	14·9	—
Cement	1·7	106	21·1	5,000
Chalk, in lumps. . .	2·0	125	17·9	—
„ powder . . .	2·64	165	13·6	—
Clay	1·9	119	18·8	—
„ fire- (Newcastle). .	2·52	157	14·2	—
Concrete	1·9	119	18·8	—
Granite, Aberdeen . .	2·62	163	13·7	10,900
„ Cornish . . .	2·66	166	13·5	6,300
Limestone, blue Lias .	2·47	154	14·5	—
„ compact . . .	2·58	161	13·8	8,000
Masonry (mean). . .	2·1	131	17·1	—
Mortar {	1·38 to 1·9	86 to 119	26 to 18·8	—
Quartz {	2·55 to 2·77	159 to 173	14·1 to 12·9	—
Sand, coarse	1·61	100	22·4	—
„ fine	1·52	95	23·6	—
Sandstone, Arbroath pavement }	2·47	155	14·4	7,884
Sandstone, Bramley Fall .	2·5	156	14·3	6,050
„ Caithness . . .	2·64	165	13·6	6,490
„ Craigleith . . .	2·45	153	14·6	5,287
„ Derby grit . . .	2·4	150	14·9	3,100
„ red (Cheshire) . .	2·15	133	16·9	2,185
„ Yorkshire paving	2·51	157	14·2	5,714
Slate, Cornish	2·51	157	14·2	10,000 to 21,000
„ Welsh	2·88	180	12·4	

FORMULÆ for the Conversion of readings of the centigrade into degrees of the Fahrenheit scale, and *vice versa*.

$$n^{\circ} \text{ C.} = \frac{9}{5} n^{\circ} + 32^{\circ} \text{ F.}$$

$$n^{\circ} \text{ F.} = \frac{5}{9} (n^{\circ} - 32^{\circ}) \text{ C.}$$

TABLE IV., showing the *Specific Gravity, &c.* of certain *Gases*.

Name of Gas	Sp. Gr. Water=1.	Weight of one cub. ft. in lbs.	Volume of one lb. in cub. ft.
Air	001293	08072	12'387
Carbonic acid . .	00197	1234	8'101
Carbonic oxide . .	00125	0784	12'753
Chlorine	00318	1968	5'081
Hydrogen	0000895	00559	178'83
Nitrogen	00125	0784	12'753
Oxygen	00143	0892	11'204
Steam	000805	0502	19'913

TABLE V., showing the *Common Percentage Composition* of certain *Alloys*.

Name of Alloy	Aluminium	Antimony	Bismuth	Cadmium	Copper	Gold	Iron	Lead	Nickel	Silver	Tin	Zinc
Aich's metal . .	—	—	—	—	60	—	1'8	—	—	—	1	38'2
Aluminium bronze . .	90	—	—	—	10	—	—	—	—	—	—	—
Arguzoid	—	—	—	—	56	—	3'5	—	13'5	—	4	23
Bell-metal	—	—	—	—	78	—	—	—	—	—	22	—
Brass—best	—	—	—	—	71'4	—	—	—	—	—	—	28'6
„ common	—	—	—	—	66'6	—	—	—	—	—	—	33'3
„ yellow	—	—	—	—	60	—	—	—	—	—	—	40
„ pin-wire	—	—	—	—	63	—	—	—	—	—	—	37
Britannia metal . .	—	6'2	—	—	1'8	—	—	—	—	—	92	—
Bronze coinage . .	—	—	—	—	95	—	—	—	—	—	4	1

TABLE V.—*continued.*

Name of Alloy	Aluminium	Antimony	Bismuth	Cadmium	Copper	Gold	Iron	Lead	Nickel	Silver	Tin	Zinc
Bronze for bearings . . .	—	—	—	—	82	—	—	—	—	—	16	2
„ „ . . .	—	—	—	—	6	—	—	—	—	—	17	77
„ for wheel boxes. . .	—	—	—	—	80	—	—	—	—	—	18	2
Dutch-metal . . .	—	—	—	—	84·6	—	—	—	—	—	—	15·4
Electrum . . .	—	—	—	—	51·6	—	—	—	25·8	—	—	22·6
Fusible metal—Lichtenberg's . . .	—	—	20	—	—	—	—	50	—	—	30	—
Fusible metal—Rose's . . .	—	—	50	—	—	—	—	25	—	—	25	—
„ Wood's . . .	—	—	50	12·5	—	—	—	25	—	—	12·5	—
German silver . . .	—	—	—	—	40 to 60	—	—	—	30 to 20	—	—	30 to 20
„ (plate) . . .	—	—	—	—	55	—	2	—	24	—	3	16
Gold coinage . . .	—	—	—	—	8 33	91·66	—	—	—	—	—	—
Gun-metal . . .	—	—	—	—	90	—	—	—	—	—	10	—
Muntz's metal . . .	—	—	—	—	60 to 64	—	—	—	—	—	—	40 to 35
Pewter, plate . . .	—	7	2	—	2	—	—	—	—	—	89	—
„ triple . . .	—	15	—	—	—	—	—	6	—	—	79	—
„ ley . . .	—	—	—	—	—	—	—	20	—	—	80	—
Silver coinage . . .	—	—	—	—	7·5	—	—	—	—	92·5	—	—
Solder—fine . . .	—	—	—	—	—	—	—	33·3	—	—	66·6	—
„ common . . .	—	—	—	—	—	—	—	50	—	—	50	—
„ coarse . . .	—	—	—	—	—	—	—	66·6	—	—	33·3	—
„ for aluminium . . .	6	—	—	—	4	—	—	—	—	—	—	90
„ brazing . . .	—	—	—	—	50	—	—	—	—	—	—	50
„ glaziers' . . .	—	—	—	—	—	—	—	25	—	—	75	—
„ gold . . .	—	—	—	—	22·2	66·6	—	—	—	11·1	—	—
„ pewterers' . . .	—	—	11·8	—	—	—	—	29·4	—	—	58·8	—
„ silver . . .	—	—	—	—	32·3	—	—	—	—	38·5	—	29·2
Speculum-metal . . .	—	—	—	—	66·6	—	—	—	—	—	33·3	—
Sterro-metal . . .	—	—	—	—	55 to 60	—	2 to 4	—	—	—	1 to 2	34 to 44
Tutenag . . .	—	—	—	—	45·7	—	—	—	17·4	—	—	36·9
Type-metal . . .	—	25	—	—	—	—	—	75	—	—	—	—
„ . . .	—	25	—	—	—	—	—	50	—	—	25	—



QUESTIONS FOR EXAMINATION.

FUEL.

1. How are the relative quantities of heat given out during the combustion of different bodies ascertained and computed?
2. Define the terms *calorific power* and *calorific intensity*, and compare carbon and hydrogen in this respect when burnt in (a) oxygen, (b) air.
3. What are the conditions favourable to a high calorific intensity?
4. Why is less heat given out by the combustion of carbon to carbonic oxide than when the latter is burnt to carbonic acid?
5. What is the effect of increased pressure during combustion?
6. Define the term *dissociation*.
7. Can the commercial value of a given coal be ascertained by analysis alone? If not, by what would you be guided in forming an opinion?
8. What is the approximate composition of (a) wood, (b) charcoal? Has charcoal any advantages over wood as a fuel?
9. State briefly the general methods of manufacturing charcoal, enumerating any special precautions which it is necessary to take.
10. What is peat? How is it prepared for use? What are its properties as a fuel?
11. Classify the various kinds of coal, and state the purposes for which each is more particularly suited.
12. In what respect does caking coal essentially differ in composition from non-caking coal?
13. Is more heat generated by the combustion of anthracite than by the combustion of caking coal?
14. What deleterious substance is always present in greater or less quantity in coal?
15. Explain and compare the most important methods of coking coal.
16. Why is coke preferable to coal for certain purposes?
17. Describe the construction of a gas-producer.
18. Describe the Siemens' regenerative system, and explain the advan-

tages and disadvantages attending the use of gaseous fuel as compared with solid fuel.

19. Sketch and describe (a) Cowper's stove, (b) Whitwell's stove.

REFRACTORY MATERIALS.

20. By what circumstances would you be guided in the selection of suitable refractory materials for the construction of the various parts of furnaces and other metallurgical appliances?

21. State what you know of the composition of clays. To what does clay owe its great value as a constructive material?

22. What is the object of mixing with raw clay used for making refractory bricks, &c. one of the following substances: burnt clay, coke dust, graphite, or silica?

23. Why is iron pyrites so objectionable in clay for crucibles?

24. Describe the manufacture of crucibles.

25. How are silicious bricks made, and for what purposes are they used?

26. What is bauxite?

27. What is dolomite, and what are its properties as a furnace material? How is it prepared for use?

GENERAL PROPERTIES OF METALS.

28. What is meant by the *tenacity* of a metal? Compare tin, steel, and copper with respect to this quality.

29. Why are metals annealed during the process of rolling them into sheets?

30. Explain why iron, though less malleable than lead, is more ductile.

31. The specific gravity of mercury is given as $13\frac{1}{2}$; what does this signify?

32. Spirit of wine burning in an earthen saucer is extinguished when poured upon an iron plate; how is this accounted for?

33. If expense were no object, what metal would be best adapted for telegraph wires?

34. Which of the metals in common use can be melted in a ladle placed upon an ordinary fire?

IRON.

35. What is *meteoric iron*? By what metal is it always accompanied?

36. Which of the ores of iron are most abundant in England and Scotland respectively? Are these the only iron ores smelted in this country?

37. Why are some iron ores calcined or roasted previously to smelting them?

38. By what considerations is the pressure of the blast from the tuyers of a blast-furnace regulated?
39. Explain the principles upon which the flux for iron ores is selected.
40. Mention the chief chemical operations which take place in the blast-furnace.
41. What is the nature of the gas issuing from the blast-furnace, and how is it turned to account?
42. Point out the chief ingredients of the blast-furnace cinder. What does a black slag indicate?
43. Name the principal constituents of cast iron, and explain their presence.
44. Explain the difference between *graphite* and *combined carbon*, and state how their proportions influence the properties of cast iron.
45. What is the effect of casting in *chills*, and how can it be accounted for?
46. By what process can hard castings be softened externally?
47. Under what circumstances does the blast-furnace yield white iron?
48. Which of the varieties of pig-iron contains the largest proportion of silicon?
49. What quality of cast iron is best suited for fine castings?
50. Why is the yield of iron from a given furnace greater in winter than in summer?
51. What is the mode of proceeding to obtain a mottled iron from a given furnace?
52. What is the effect of phosphorus on cast iron?
53. Describe a cupola furnace, and its use.
54. For what purpose is lime introduced into a cupola furnace? What evil results from using an excess of lime?
55. State how chilling may be avoided when casting in sand.
56. Describe the *finery-hearth*.
57. How is the composition of pig-iron affected by the process of refining?
58. Compare the composition of finery cinder with that of blast-furnace cinder.
59. Explain the principle of the puddling process.
60. Mention the essential particulars in the construction of a puddling furnace.
61. Point out the difference between *dry puddling* and *pig-boiling*.
62. State the composition of tap-cinder.
63. Mention the principal defects of the puddling process.
64. What are the characters of mill bar, and how is it converted into merchant and best bar?
65. In what particulars does the reheating furnace differ from other furnaces?

66. What explanation can be given of the improvement of bar iron by piling?
67. Explain the meaning of the terms *hot-short* and *cold-short*, as applied to iron.
68. By what physical properties is steel distinguished from iron?
69. Enumerate the most essential points to be attended to in the manufacture of puddled steel.
70. Point out the best materials for making permanent and temporary magnets respectively.
71. How may iron bars be converted into steel?
72. State the peculiar features of the cementation furnace.
73. What is the influence of the duration of the cementing process upon the character of the steel?
74. Point out the defects of blister steel, and describe its conversion into shear steel.
75. How is cast steel produced? What advantages are secured by adding Spiegel-eisen to it?
76. Is any chemical difference observable between hard and soft steel?
77. By what process can the highest degree of hardness be conferred upon steel?
78. Why is oil sometimes employed for hardening steel?
79. Describe the process for reducing the hardness of steel.
80. What is the object in bluing steel articles, and how is it effected?
81. Describe and explain the process of case-hardening.
82. What descriptions of articles are usually case-hardened?
83. By what mode of treatment may small castings be converted into malleable iron?
84. What is meant by the *Catalan process* for the extraction of iron? Why is it not practised in England?
85. How would you explain the carburisation of the iron in the cementation process?

BESSEMER PROCESS.

86. Describe and illustrate by sketches the apparatus necessary in the Bessemer process.
87. What means have been devised to enable the converter to be rapidly changed? What advantage is thus gained?
88. What kind of pig-iron would you use for this process? Give your reasons.
89. Explain how the process is carried out.
90. In what respect does the Thomas-Gilchrist or 'basic' Bessemer process differ from the Bessemer or 'acid' process: (a) in the nature of the plant; (b) in the pig-iron used; (c) in the carrying out of the process; (d) in the character of the product?

SIEMENS PROCESS, &c.

91. Describe the arrangements necessary to effect the decarburisation of pig-iron in a reverberatory furnace and to maintain the decarburised metal in a state of fusion.

92. Describe the two methods in general use for the manufacture of mild steel or ingot iron in a Siemens furnace.

93. Why is it necessary to use Spiegel-eisen or ferro-manganese in this process? Under what circumstances is ferro-manganese to be preferred to Spiegel-eisen?

94. Explain the arrangement of (a) the Pernot furnace; (b) the Ponsard furnace. What are the probable advantages and disadvantages of these furnaces?

95. What is the effect of the following elements on iron, individually, and also in the presence of one another: carbon, silicon, phosphorus, sulphur, copper, chromium, and tungsten?

COPPER.

96. Where is copper chiefly found in the metallic state?

97. Give the composition of the most abundant of the English ores of copper.

98. Name the chief seat of British copper smelting, and state the reasons for its selection.

99. What is the composition of fluor spar, and why is it useful in copper smelting?

100. Enumerate the chief stages of the process for smelting copper ores, with the objects to be attained by each.

101. Explain the management of the clinker grate in the roasting-furnace.

102. What is the nature of *copper-smoke*?

103. Point out the chief differences between metal slag and ore-furnace slag.

104. What advantage is secured by air-channels in the fire-bridge of the roasting-furnace?

105. Give an explanation of the terms *underpoled* and *overpoled* copper.

106. What is the reason for scorifying copper with lead before rolling?

107. Why does the process adopted at Mansfeld differ from the Welsh process of copper smelting?

108. Describe the process for refining the rosette copper at Mansfeld.

109. How is cement copper obtained, and what is its quality?

110. What is bean-shot copper used for, and how is it prepared?

111. Is impure copper more inclined to hot-shortness or to cold-shortness?

112. Which of the uses of copper is most seriously interfered with by the presence of small quantities of foreign matter?

113. Describe the process of obtaining *precipitate* copper from Spanish pyrites. How is the silver recovered?

TIN.

114. What is the composition of tinstone, and where is it chiefly found?
115. Explain the meaning of *stream-tin*.
116. Give an outline of the mechanical treatment of tin ores.
117. Describe the smelting process, pointing out the precautions against loss of tin in the slag.
118. How is tin refined by liquation?
119. What are the objects of *boiling* and *tossing*?
120. How does the process followed in the Saxon tin-works differ from the English process?
121. What useful compound is prepared from the wolfram occasionally associated with tin ores?
122. By what physical peculiarity may tin be immediately recognised?
123. How is tin-plate made?
124. Point out the difference between charcoal-plate and coke-plate.
125. Why is a kettle of block-tin so much more expensive than a common tin kettle?
126. How is copper cleaned previously to tinning it?
127. Should alloys be regarded as merely mechanical mixtures, or as chemical compounds?
128. Mention the chief useful alloys of copper and tin.
129. State the composition of the current bronze coin.
130. What is the method of hardening and tempering bronze?

ZINC.

131. Enumerate the chief ores of zinc, stating their composition.
132. In what respect does the process of extracting zinc differ from those employed for most metals?
133. What is the commercial name for zinc? Is any other substance ever called by it?
134. By what rather rare metal is zinc usually accompanied in its ore?
135. Under what conditions may zinc be rolled into sheets?
136. How can zinc be purified from lead?
137. What is meant by *burned zinc*?
138. *Corrugated iron* is used for building; what is it?
139. What impurity is especially objectionable in the zinc employed for making brass?
140. State the composition of Dutch leaf gold.
141. For what reason is a little lead sometimes added to brass?
142. How is brass lacquered?

- 143. By what process are pins coated with tin?
- 144. State the composition of sterro-metal. For what purpose has it been employed?
- 145. What is aluminium-bronze?

NICKEL AND COBALT.

- 146. Describe the physical properties of these metals.
- 147. Which are the principal ores of nickel and cobalt, and what is their usual composition?
- 148. Give the principle of the methods employed in extracting nickel and cobalt from their ores.
- 149. For what purposes are these metals useful in the arts?
- 150. By what means are nickel and cobalt rendered malleable?

LEAD.

- 151. Give the name and composition of the most abundant ore of lead.
- 152. What other metals are generally found in lead ore?
- 153. Point out the peculiar features of the reverberatory furnace for lead-smelting.
- 154. In what respect does the process of smelting galena resemble that adopted for copper pyrites?
- 155. Why is iron sometimes added to the charge in the smelting-furnace?
- 156. In what respect does the economico-furnace differ from the reverberatory furnace for smelting lead ores?
- 157. What is the nature of the improving process?
- 158. How may a poor argentiferous lead be rendered fit for cupellation?
- 159. Point out the distinction between the high and low systems of concentrating silver-lead.
- 160. By what method can leaden pipes be protected from the corrosive action of water?
- 161. Of what material are metallic pencils composed?
- 162. State the composition of type-metal.
- 163. Why does antimony unfit lead for rifle-bullets?
- 164. How is the spherical form given to small shot?
- 165. Describe the mode of constructing the leaden chambers for the vitriol-works.
- 166. What is the nature of the change by which lead is converted into an earthy mass, easily crumbling?
- 167. State the composition of pewter. Why is the standard proportion of lead fixed at one-fifth only?
- 168. What is the difference between pewterers' solder and common solder?

169. Why is zinc dissolved in the muriatic acid which is used for soldering?
 170. Give the composition of hard solder. What is the effect of adding silver to it?

SILVER.

171. Explain the principles of the process of cupellation.
 172. What is litharge? How is it obtained?
 173. Why does melted silver sprout in solidifying?
 174. Mention the principal differences between the English and German cupellation processes.
 175. Describe the extraction of silver from the black copper at Mansfeld.
 176. What is the principle of the extraction of silver by amalgamation?
 177. Why is this process carried on so differently in Mexico and Saxony?
 178. Describe a process in which common salt is employed for extracting silver from the ore.
 179. What is the composition of silver coin? Why is not pure silver circulated?
 180. What is understood by the assayer's statement that a sample of silver is *worse 5 dwts.*?
 181. How can the genuine character of a shilling be tested without injuring it?
 182. Why does silver tarnish so quickly where coal-gas is employed? How may it be cleaned?
 183. Point out the difference between old silver plate and electro-plate.
 184. What solution of silver is best adapted for electro-plating?

GOLD.

185. In what respect does the mode of occurrence of gold in nature differ from that of most other metals?
 186. Describe the washing of auriferous sand in the cradle.
 187. How is gold extracted from pyrites in the Tyrol?
 188. Describe the process of parting by sulphuric acid. How is the silver recovered in the metallic form?
 189. Is native gold absolutely pure?
 190. What is meant by *fine gold*, and by *gold of 18 carats fine*?
 191. By what rough test is gold commonly tried? Show where it is fallacious.
 192. Explain the use of the touchstone.
 193. What is meant by *inquartation* in the assay of gold, and why is it necessary?
 194. Describe the manufacture of gold-leaf.
 195. How are buttons coated with gold?

MERCURY.

196. In what forms is mercury met with in the mineral kingdom?
197. What is the nature of the process employed for extracting mercury at the works in Idria?
198. Is there any more economical process than this?
199. Give a ready method for ascertaining the purity of quicksilver.
200. Why is mercury selected for filling thermometers?
201. What is an *amalgam*?
202. How are looking-glasses usually made?

PLATINUM.

203. What is the condition of platinum in the ore?
204. Point out the chief differences between the old and new processes for treating platinum ores.
205. How is platinum melted in considerable quantities?
206. In what branch of manufacture is platinum largely employed?
207. Why should platinum vessels never be heated in a coal fire?

ANTIMONY, &c.

208. What is the difference between crude antimony and regulus of antimony?
209. By what peculiarities is antimony distinguished from all other metals?
210. What is remarkable in the process of extracting bismuth from its ores?
211. Mention the impurities commonly found in the bismuth of commerce.
212. Upon what peculiarities do the uses of bismuth depend?
213. Where is aluminium met with? Does its high price depend upon its scarcity?
214. What useful applications have been found for aluminium?
215. From what circumstance does magnesium derive importance?

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